

Metallurgical *and* Chemical Engineering

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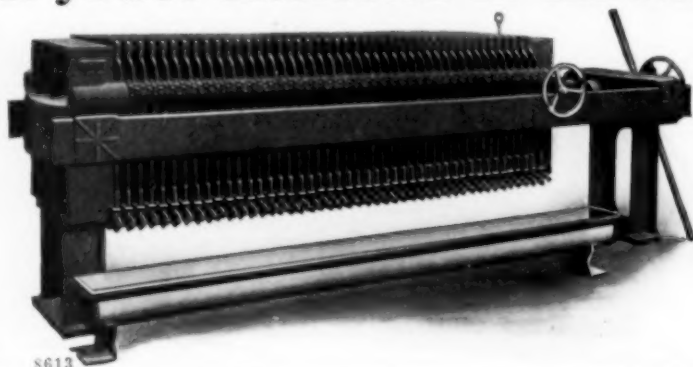
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The Atomic Weight of Gold

At the recent Leipzig convention of the German Bunsen Society, Dr. Krey in his speech of welcome on behalf of the German Chemical Society made the characteristic remark that not so long ago the older generation of orthodox chemists used to look at physical chemistry with feelings very much like those with which we (or some of us) view the art works of futurists and cubists. It is true this sentiment has changed, but if orthodox chemists were present at the Leipzig meeting, we are sure they must have been shocked by the papers by Dr. K. Fajans and Dr. O. Hönigschmid, who spoke of an indefiniteness of atomic weights and backed it up by the example of lead. And now they have been confirmed by so eminent and conservative an authority as Prof. Theodore W. Richards, of Harvard, who together with Max E. Lambert has determined the equivalent weights of sixteen samples of lead chloride obtained from different sources. "The result is amazing. Evidently, then, the chemical equivalents of these different specimens are markedly different from one another." Lead of radioactive origin has a lower atomic weight than common lead. And yet all the different specimens are chemically identical according to the most refined Harvard methods (we apologize to every Harvard man for this tautology). They are even identical according to spectroscopic analysis. They only differ in radioactive properties, and—in atomic weights. But what does it prove? That atomic weights are no longer constants of nature? Hardly, but it suggests that in lead there may be some admixture of very remarkable properties. What is it? Here is a wonderful opportunity for experimental research.

Raw Material for Steel Production

A striking feature of the statistics of steel production in the United States in 1913, just issued by the Bureau of Statistics of the Institute, is that they show an increase in the output of ingots and castings of only 49,571 tons, or one-sixth of one per cent, over the preceding year, whereas pig iron production increased 1,239,215 tons, or 4.2 per cent. It is impossible to account for this divergence by inferring that stocks of pig iron increased during 1913. Analysis indicates clearly that the divergence was largely due to there being a higher proportion of basic pig iron to basic open-hearth steel in 1913 than in 1912, and this fact suggests a very interesting trend. The production statistics are as follows:

	1912	1913
Pig iron in gross tons.....	29,726,937	30,966,152
Steel ingots and castings..	31,251,303	31,300,874

The production of foundry and mill grades, including foundry, malleable and forge iron, etc., increased from 6,423,312 tons to 6,605,366 tons, or 182,024 tons, 2.8 per cent, quite in keeping with the increase in pig iron as a whole. The production of Bessemer and low-phosphorus iron decreased from 11,664,015 tons to 11,593,385 tons, or 70,630 tons, while the production of Bessemer and acid open-hearth steel, almost the sole outlet for Bessemer and low-phosphorus pig iron, decreased from 11,467,122 tons to 10,801,011 tons, or 666,111 tons. The excess of pig iron over steel in 1913 was 7.3 per cent, which is only a trifle above the average; the excess in 1912, on the other hand, was below the normal, being only 1.7 per cent. It will be recognized that these percentages do not represent the loss in converting Bessemer pig iron into Bessemer steel ingots. There are other factors, the consumption of Bessemer pig iron in foundry operations, which is, however, quite limited, and, on the other hand, the use of scrap in acid open-hearth steel practice.

Comparing the production of basic pig iron with the production of basic open-hearth steel ingots and castings shows that in 1912 the proportion of pig iron to steel was a trifle over 58 per cent, while in 1913 it was nearly 62 per cent. This proportion is always suggestive of the quantity of scrap, chiefly new scrap, used in the basic open-hearth furnace in conjunction with pig iron, and we give below the proportions for each year since the statistics were first gathered. The statement begins with 1896, when the output was only 336,403 tons of basic pig iron and 776,256 tons of basic open-hearth steel, and ends with 1913, when the output was 12,536,693 tons of basic pig iron and 20,344,626 tons of basic open-hearth steel.

PROPORTIONS OF BASIC PIG IRON TO BASIC OPEN-HEARTH STEEL INGOTS AND CASTINGS, PER CENT

1896, 43.3	1905, 52.5
1897, 52.7	1906, 52.0
1898, 50.1	1907, 52.4
1899, 47.4	1908, 57.1
1900, 42.1	1909, 61.5
1901, 40.0	1910, 59.5
1902, 45.2	1911, 58.0
1903, 43.0	1912, 58.1
1904, 48.5	1913, 61.6

Minor variations in the percentage from one year to another may possibly be due to fluctuations in the volume of stocks, but these stocks are never particularly large, and the table shows conclusively a definite and important trend. In the earliest years the industry had not found its gait, and although there was doubtless much scrap available the largest proportion was not used. The maximum employment of scrap appears to have been in 1901, when the pig iron production was only 40 per cent of the steel production. For four years ending with 1903 the proportion of pig iron averaged 42.6 per cent, the proportion rising rapidly thereafter until 61.5 per cent was reached in 1909, and then with slightly lower percentages for three years a new high point of 61.6 per cent was reached in 1913.

The decrease in the relative consumption of scrap is not to be attributed to a failure in the supply of old material. Doubtless the outcome of old material increases from year to year, with more or less reference to the fact that ten, fifteen or twenty years ago, whenever it was that the present old material was new material, just put into use, the production and consumption of steel was increasing at a rapid rate.

Old material, however, does not furnish the chief supply of scrap for the open-hearth steel furnace, for the production of new scrap, including casting pit scrap, scrap produced by cropping in rolling, the shearing of plates and the fabrication of various steel mill products into final form, is much larger than the outcome of old material. The Bessemer steel process does not use its own scrap, and has not done so to any extent since the advent of the basic open-hearth furnace. The Bessemer steel industry, however, is decadent.

It does not follow that there has been in all plants an equal decrease in the proportion of scrap charged. The proportions have for years varied widely in different classes of plants. Those which use the least scrap, scarcely any, are, of course, the duplex works, partially treating the molten iron in the converter and finishing it in the basic open-hearth furnace. Statistics of duplex steel are available only for 1912 and 1913, the production being respectively 1,438,654 tons and 2,210,718 tons. Assuming for the computation that these duplex works used an equal tonnage of basic pig iron, it develops that the remainder of the industry used 54.8 per cent of pig iron in 1912 and 56.9 per cent in 1913. If one were able to segregate further and allow for the operation of Talbot furnaces, the proportion of pig iron would be found to be further reduced, but there is little doubt that the figures would show an increase in 1913 over 1912.

It is easy to foresee that as the industry grows in future and the basic open-hearth preponderates still more greatly over the Bessemer there will be a continued increase in the use of what may be called the non-scrap processes, and an increase in the proportion of pig iron used at the works which practice the regular pig and scrap process. The day of the blast furnace is not over, and it is even a question whether there is to-day sufficient blast furnace capacity at steel works to supply pig iron for full operation of the basic open-hearth capacity, increased as it has been so largely in the past few years.

Arms and the Man

The past is effete and from us. The present is here and of us. The future is to come and for us. The past is valuable only as it teaches. The present is valuable only as we act. The future is good in that we can see. The three simple words, see, think, and act are part of a man's character, and how well one "sees, thinks and acts" distinguishes a man from a "mutt." Now coming from the past we find the adjectives noble, knightly, chivalrous that today possess the connotation of ro-

mance and give a glorious sheen to our thoughts. There is little of much greater value, nothing much more soul-satisfying, than the glory that comes from the day when knighthood was in flower.

How did these words acquire so much of beauty and of poetry and of spiritual grace, and how did knights rise to an honorable position? How did Chevalier Bayard "*sans peur et sans reproche*" reach the ideal position? The reason we think can be found in progress of metallurgy and the reactance of metallurgy on esthetic and moral values is seen once again.

Brushing aside all verbiage and proceeding directly to our theme, we find that the man from prehistoric cave-man, seen incarnate today in a literary fashion in Jack London, has ever tried to get perfect weapons of offense and defense. This procedure was started in history by an eminent metallurgist bearing initials T. C. and went along under sign of the Hephaistos to the days of Colt, Krupp, and Maxim. It was by metallurgical reactions and by metallurgical endeavor that the suit of mail of Milan and the Toledo blade became perfect till at last even the feet were covered with protectors called "*sollerets*."

The perfection of armor was attained in the 15th century and was of such a degree that the shield was discarded as useless and the armored knight who was strong to mount his horse and move his muscles could not be wounded at all. The process was a gradual evolution, and had, long before that time in 1000 A.D. reached a temporary ultimate as is marked by the Battle of Hastings, where Norman knights conquered the Saxon warriors because they were better armed. Probably for six or seven centuries the man who had armor and was strong enough to carry it successfully was the first-class fighting man who dominated things so far as his power could carry. Finally, in the Battle of Crecy, which marked the introduction of gunpowder as a practical military agent, the kibosh was put on the knights and a new order of social affairs was started.

Let us analyze carefully these social changes—we had almost said economic changes, but all we had described happened before the days of Adam Smith. In feudal times a suit of armor was a valuable possession and constituted capital for its owner. But it weighed as much as 150 pounds and there were only a few men who were strong enough to carry such a weight and fight effectively. It was the exceptionally robust and tough man who became a knight. But provided he reached such a point in physical stamina that he could carry the armor and have surplus strength to use in offense, he was invincible against the man who did not possess armor or was not strong enough to fight in it. Given a man who reached a certain point in his physical make-up, he became all powerful. Below that point a man was defenseless because he was too slow.

For the men of the lower degree of fighting quality the only safety was in flight, and once caught whence he could not flee, he was lost and became a social by-product.

Such is in crude and bold outline the "*Ursache*" of knighthood. The knight was the exceptional man who

was strong enough to carry armor and who, therefore, became militarily far stronger. A knight in armor could defeat in open fight, "*ceteris paribus*," 30 other men without armor.

Now with such objective conditions we might surmise that subjective conditions would be bad, that cruelty would be synonymous with bravery, that bad faith and treachery would constitute success, and that the weak would be pillaged by the strong to extermination. It would seem that the wolf and the shark would have assumed human form by a practical form of transmigration of souls.

What actually happened was quite the reverse. While injustice and rapine did exist, they decreased rather than increased. Loyalty, fealty, honor and self-sacrifice were in the knights' code given as the ultimate spiritual values. Women were placed in an idealized state and the flavor of chivalry pervaded life. The strong protected the weak and the weak repaid the strong by faithful service. In short, the power of association prevailed over the power of dissociation in a period when the "*a priori*" philosopher would have said that it was a question of how long before the "*dog-eat-dog*" method would have exterminated mankind. But while there was much of badness, badness did not increase, but rather goodness. Humanity energized joyfully in the days of old when knights were bold and ladies were fair. This is seen in the beautiful legends and stories of Arthur and the Troubadors. The novels of Scott and Dumas are true to the life of the feudal times. It was undeniably a time when moral values and spiritual values grew in beauty and strength.

With such cogitations and with such reflections in mind, we are struck with a certain analogy that today obtains in the world. The weapons that metallurgical science has forged today are commercial weapons of offense and defense, new processes to fight with, new thoughts to think with—science to materialize and to practicalize. But these weapons are so heavy that only the exceptional man can wield them. The thoughts of modern science are of commercial value, but it needs a thinker to absorb them. Those of lesser mentality who try, fail because of the nervous worry induced. Their powers are not strong enough and so they must sink to a subordinate position just as many men, weak muscularly, became in olden times archers, esquires or men-at-arms. We can see evidences of the inception of a process similar to the process that developed feudalism and chivalry. It may be undemocratic to foresee a triumph of aristocracy, but the future aristocracy will be founded on spiritual values and so an enduring one. The efficient man is vastly more efficient to-day because he uses his senses and thoughts in a transcendental manner for practical purposes.

We may be bold to predict from the foregoing that out of the present chaos that calls itself democracy a system will arise in which faithfulness, honor, loyalty and truth will be the goals, that the strong will protect the weak, that things will synergize and harmonize, so that the power of darkness will be overcome. Such a conclusion seems as comforting as it is irresistible.

Readers' Views and Comments

Igneous Concentration of Metallic Values from Mixed Sulphides

To the Editor of Metallurgical & Chemical Engineering:

Sir:—It is not easy to find an excuse for arithmetical errors such as occur in my paper on Igneous Concentration of Metallic Values from Mixed Sulphides, pub-

lished in your journal, May, 1914, page 307. When such errors are found, the most fitting apology is a correction. Will you help me apologize to your readers by printing the corrected tables?

In a certain sense these errors are not important, as I was more concerned with the greater importance of utilizing as fully as possible the fuel values of various constituents of mixed sulphides, than with the publication of exact values in unfamiliar units.

The tables were compiled under conditions which prevented me from checking the results in the customary way.

F. L. CLERC.

Estes Park, Col.

Is There a Complex Ore Problem?

To the Editor of Metallurgical and Chemical Engineering:

Sir:—On reading Mr. Lewis B. Skinners' contribution to the discussion on the above subject in your June issue, I was much surprised to find the following statement: "In passing, as there are fads in metallurgical lines as well as in others, I might note that we are going through a stage when the whole fraternity is on the *qui vive* trying to make every ore amenable to treatment by some flotation process. In the case of one-mineral separation in the United States this process has proven an aid; but it is of no benefit in the case of ores of zinc, which are truly complex, although being advocated for such. I do not know of any complex ore in Colorado, for instance, that could possibly show any advantage under this treatment."

It may surprise Mr. Skinner to hear that no fewer than four flotation processes, viz.: Potter, Delprat, Minerals Separation, and DeBavary, were invented for the express purpose of treating complex zinc ore at Broken Hill, Australia, and that during the last ten years several million tons of zinc-lead-silver ore and tailings have been profitably treated by these processes. As I was employed in mills using these processes at Broken Hill eight years ago I know whereof I speak.

I would like to see some of that "complex ore in Colorado that could not possibly show any advantage under this treatment." It might be worth while to have some experiments made with it.

Enormous strides have lately been made in "selective flotation" at Broken Hill (apart from the Horwood process), about which nothing has been published, and about which little or nothing is known in this country.

I might mention that the Murex process (not strictly a flotation process) is treating very complex zinciferous slime at the State mines in Germany.

W. MOTHERWELL.

Humboldt, Arizona.

* * *

To the Editor of Metallurgical & Chemical Engineering:

SIR:—Having followed with great interest the discussion of the complex-ore problem in recent numbers of your magazine I feel impelled to add my mite by way of comment on a few points.

1. Is there a complex ore? Or, going for instruction to the English dictionary, is there an ore "composed of several parts or components; knit, woven or folded together; involved, complicated, intricate; an aggregation; a number of metals combined into one grain or crystal?" All agree that there is. The question in my mind is: Is there a single ore that is not complex?

TABLE I

COMBINED WITH		GIVES	AND EVOLVES
One Lb. of	Lb. of	Lb. of	B.T.U.
Zinc	Oxygen 0.246	ZnO 1.246	2,349
Iron	Oxygen 0.286	FeO 1.286	2,111
Copper	Oxygen 0.252	Cu ₂ O 1.252	1,067
Copper	Oxygen 0.126	CuO 1.126	6,197
Carbon	Oxygen 1.333	CO 2.333	4,374
Carbon	Oxygen 2.667	CO ₂ 3.667	14,580
Hydrogen	Oxygen 8.000	H ₂ O 9.000	52,254
Sulphur	Oxygen 1.000	SO ₂ 2.000	3,895
Oxygen	Zn 4.062	ZnO 5.062	9,540
Oxygen	Fe 3.500	FeO 4.500	7,391
Oxygen	Cu 3.975	CuO 4.975	4,041
Oxygen	Cu 7.950	Cu ₂ O 8.950	4,929
Oxygen	C 0.750	CO 1.750	3,280
Oxygen	C 0.375	CO ₂ 1.375	5,464
Oxygen	S 1	SO ₂ 2.000	3,895
Oxygen	H 0.125	H ₂ O 1.125	6,532
Iron	S 0.554	FeS 1.554	771
Zinc	S 0.492	ZnS 1.492	1,191
Copper	S 0.503	CuS 1.503	286
Sulphur	Zn 2.031	ZnS 3.031	2,419
Sulphur	Fe 1.750	FeS 2.750	1,350
Sulphur	Cu 1.988	CuS 2.988	569
Sulphur	Cu 3.975	Cu ₂ S 4.975	1,142

TABLE II

COMBINED WITH		GIVES	AND EVOLVES	ALSO GIVES	EVOLVES	TOTAL	ABSORBS	NET
One Lb. of	Lb. of	Lb. of	B.T.U.	Lb. of	B.T.U.	B.T.U.	B.T.U.	B.T.U.
ZnS	O 0.495	ZnO 0.835	1574	SO ₂ 0.66	1285	2859	798	2061
FeS	O 0.650	FeO 0.818	1343	SO ₂ 0.728	1418	2761	491	2270

To burn completely 1 lb. ZnS, requires 0.495 lb. of oxygen for zinc and sulphur, and develops 2061 B.T.U. of available heat.

To burn completely 1 lb. FeS, requires 0.650 lb. of oxygen for iron and sulphur, and develops 2270 B.T.U. of available heat.

To burn 1 lb. of carbon to CO₂ requires 2.667 lb. of oxygen and develops 14,580 B.T.U. of available heat.

To burn 1 lb. of carbon to CO requires 1.334 lb. of oxygen and develops 4374 B.T.U. of available heat.

It then requires about 7 lb. of ZnS to replace 1 lb. of pure carbon burnt to CO₂, but only about 2 1/8 lb. if burnt to CO.

Heat developed per lb. of oxygen for ZnS is 2061 = 4164 B.T.U.

Heat developed per lb. of oxygen for FeS is 0.495 = 2270 = 3492 B.T.U.

Heat developed per lb. of oxygen for C to CO₂ is 0.650 = 14,580 = 5467 B.T.U.

Heat developed per lb. of oxygen for C to CO is 2.667 = 4374 = 3280 B.T.U.

Heat developed per lb. of oxygen for C to CO is 1.334

Is there a complex-ore problem? For more than thirty years of practice, study and experiment in the treatment of refractory ores, mainly by concentration, I have faced that problem. It is ever with us here in Colorado. We are getting more light, helps—among which such discussions as this are valuable—improvements in methods and machines; yet the problem is still here, complex indeed, inasmuch as almost every individual ore body calls for a separate solution.

Not only are many ores complex, but they call for a complexity of conditions in their treatment. I have found minerals of five different specific gravities in one ore on one table.

2. A great improvement might be made by employing or educating skilled operators. We have too few operators who are students, working with heart and mind for the success of the mill rather than for tally time and pay day. How shall we get them? Where shall we make them? That is a problem also, or perhaps a different phase of the same problem of complex-ore treatment.

3. Undoubtedly there are many mines now idle which would pay under careful and skilful management. If we would systematize our mills better, cut out the idea of "foolproof" machinery and employ at a premium a few interested, skilled and studious men who will work not only for salary but for success; cut out the idea of a big tonnage as the first essential (especially where it is impossible to get it with existing capacity of mine and mill) and hold to the idea of closest saving of mineral values and economy in all lines, we should have less idle mines, less decaying mills, monuments of failure and loss.

4. As to the element of complexity in different ores, let us consider a body of tellurium, for example. Good concentration will recover a fair percentage of values. Some of the tellurium will float and may be recovered by a method of flotation. Other values are like clouds of smoke and here cyanide gets in its work, providing the ore is oxydized or amenable to cyanide treatment. Occasionally there are particles of free gold and to save this we must amalgamate, concentrate or blanket—perhaps all three. We find pure gold bound up in grains of quartz so fine that after passing through a screen 100 meshes fine they still defy cyanide to penetrate and find their value; yet here good concentration by use of slimers and carpets will do much. The use of the glass to detect the losses is very essential. Then in many telluride ores very fine sulphides carrying high values are bound up in apparent gangue, calling for fine grinding and pebble mills and possibly oil flotation. Have we not here a complex condition, a complexity of methods, a complex problem calling for skilful intelligent handling?

5. A lead-iron-zinc-sulphide ore is, of course, complex enough to start with. Where a good concentrator will make good smelting values and at the same time cut out a middling product of iron cubes containing some zinc and traces of lead too low to pay its way, a late cyanide process has extracted 87½ per cent of the values with a 2½ per cent solution. I believe it would work on electrostatic iron as well.

An iron and free milling ore contains grains of quartz whose value will sometimes exceed the value of the heads. These can be cut out by a good concentrator and reground, occasionally in small mills and continuously in large ones, with good results.

When we go into the milling of refractory ores with knowledge of the problem before us and with a definite aim and plan of procedure, when we throw prejudice aside, when we make careful preliminary tests before installation and follow the test lines, when we employ conscientious and capable engineers who know what is

required and are as anxious for our success as for our money, when we pick out efficient men for operators and give them the support that Carnegie gave and Edison gives to his men, then we shall begin to solve the problem.

6. The use of the microscope seems to me indispensable in the concentrating mill. An operator who is interested in his work and a student should be the possessor of the best pocket glass to be had. It will keep him posted as to many things. With it he can determine how clean his tailing sands are from attached mineral, with it he can be sure of the product he is making. In fact, every mill foreman should have a glass, a buck-board, pan and screens at hand for occasional pan and grinding tests, to be followed by assays, that he may know just what he is doing. But the first duty is to save the mineral which is freed, and until those "higher up" are interested to correct faults in the primary work the man running his shift is not likely to bother much about results.

Many pages might be written on the questions outlined and facts stated as to actual practice which are unbelievable, but the subject has already been thoroughly and ably handled. Such inquiries and discussions are in the right line. We hope to have more. We are in need of an army of thinkers and students to work out the many sides of the problem.

IRA F. MONELL.

Boulder, Colorado.

Heavy Mexican Oil as Fuel

To the Editor of Metallurgical & Chemical Engineering:

SIR:—Mr. R. M. Chatterton's recent communication as set forth on your pages 364-5 of your June issue is most interesting and helpful to those making power for electrochemical purposes.

Concerning my comments before the American Electrochemical Society, etc., it is not the possibility of using oil engines to operate on heavy Mexican fuel oil that I brought forth, but rather that such is now the fact and has been for some years past. It is true that Diesel oil engines in Texas do not operate successfully on heavy Mexican oil 18 deg. Beaumé or less, and it is well known that to operate well the Diesel engine must utilize the higher-grade fuel oils. There is, however, an improvement on the Diesel oil engine which has all its advantages, namely, high economy, less attendance than a coal power plant, etc., and which is burning these heavy Huasteca oils containing as much as 60 per cent asphaltum, and doing it successfully. A 1000-kw plant of these heavy Mexican oil-burning engines is in operation.

Records taken from one of these plants and comparison with Mr. Chatterton's gas-power plant for the same load-factor, 0.73, are here recorded. The crude oil costing 2.68 cents per gallon.

	Cost per Kw-hr.			
	Coal or oil fuel	Attendance	Supplies	Fixed charges
Gas power plant..	0.34	0.266	0.0282	0.470
Oil engine plant..	0.177	0.175	0.0282	0.355
				1.104 cents
				0.733 "

From the above it is evident that crude oil for fuel for the oil engine would cost 3.75 cents per gallon before it would equal the cost of the producer gas equivalent with coal at \$3.50 a ton. As there is no producer, and no manual firing to be done, the labor is decreased by one-third. The supplies are about the same. Fixed charges are less, as the initial cost is lower, space occupied is less, and there is no producer to deteriorate. In fact, the engine and a tank of oil being practically the entire power plant. The oil engine referred to is the De La Vergne.

New York City.

C. H. VOM BAUR.

The Western Metallurgical Field

Smelting Furnace for Cyanide Precipitate

In a paper presented at a recent meeting of the Columbia local section of the American Institute of Mining Engineers, held at the State College of Washington, Pullman, Washington, Mr. P. S. Anderson, of Baker, Oregon, submitted some notes on a smelting furnace for cyanide precipitate. He anticipated that the matter would be of interest to operators of small cyanide plants where the installation of more expensive tilting furnaces is not practical. The furnace is described as being well adapted to the Taverner lead-smelting process, which the author had used for many years with good results. The last furnace was in use nearly four years. It was built in a car body, dimensions 24 in. by 42 in., and the brick work had to be replaced about every twelve months. Mr. Anderson's notes briefly described the construction and operation. The furnace is charged with about 100 lb. of cyanide precipitate, litharge and flux, approximately 70 lb. damp precipitate being used. As this melts down, more is added until there is about 7 in. of molten material in the furnace. After stirring, and when the slag is fluid, the first tapping of slag is taken off if necessary; otherwise, scrap consisting of old cupels, speiss, etc., is added, and fusion completed. In tapping, about two-thirds of the slag is taken off in the mold, and the remainder of the charge is heated until the cupel, on testing with a rod, feels smooth. Then the entire remainder of the charge is poured into a mold. After cooling, slag and speiss are removed, and the lead bar is placed in the furnace for cupelation.

Although lead is allowed to go to waste, there is a compensating advantage in the fact that slag and bullion are cleaner. Very little gold is found in the flue-dust, although there is considerable silver. Samples give average results of 12.10 oz. gold and 1140 oz. silver per ton. Slags assay, on an average, two cents per pound in gold and silver. Gasoline consumption is about one gallon per 18 lb. of precipitate, including cupelation and resmelting.

Oil-Fired Blast Furnace in Washington

An oil-fired blast furnace has been erected at the Copper King mine, Chewelah, Washington, by J. J. Anderson, patentee. A description of this type of furnace and some data on experimental operation were given in this journal, September, 1912, page 543. It is believed that this is the first furnace of the kind installed in the Northwest for smelting copper ore on a commercial scale. For some time experiments were made at Van Anda, Texada Island, British Columbia, under the metallurgical direction of Mr. Thomas Kidzie, formerly superintendent of the Northport Smelting & Refining Company's works at Northport, Washington. At his suggestion Mr. Anderson made improvements in the furnace, based on the experience gained in the trial demonstrations at Van Anda. Several runs were made, of a few days each.

The present installation at Chewelah is expected to be in continuous operation, and the results will be looked for with much interest. The capacity of the furnace is said to be about 100 tons per day. As the United Copper Company is reported to have more than 7000 tons of 5 per cent copper ore ready for smelting, there will be no lack of suitable material and the furnace will be given a fair trial.

Motherlode Sheep Creek Mill, British Columbia

At a recent meeting of the Western branch of the Canadian Mining Institute, Mr. W. P. Alderson, general manager of the Motherlode Sheep Creek Mining

Company, presented some notes on the operation of the company's mill, from which the following data are taken. The ore consists of partly oxidized quartz and quartzite and is treated by an oil-sliming cyanide process. The mill is equipped with 10 stamps and Merrill cyaniding apparatus. Power is generated by six Pelton wheels. The following figures have reference to a three-month period.

Tons of ore milled.....	8781
Average value per day.....	\$13.37
Tons ore milled per day.....	98.3
Tons milled per stamp per day.....	9.83
Percentage of time run.....	98.1
Pebble consumption per ton ore milled, lb.....	4.84
Cost of pebbles per ton ore milled.....	\$0.08
Extraction by amalgamation, per cent.....	72.1
Total extraction, per cent.....	97.1
Precipitation:	
Tons solution precipitated.....	14,957
Tons solution precipitated per ton ore.....	1.7
Value solution, per ton.....	\$1.97
Tailing:	
Total value.....	\$3,336.78
Average value per ton.....	\$0.38
Chemical consumption per ton ore milled:	
Potassium cyanide, 98 per cent, lb.....	1.84
Lime.....	2.17
Zinc.....	0.31
Total cost of chemicals.....	\$0.38
Cost of milling, per ton:	
Labor.....	\$0.515
Supplies.....	0.674
Sundries.....	0.212
Total.....	1.401

Chlorination at Park City, Utah

Since publishing a description of the metallurgical work of the Mines Operating Company, Park City, Utah, in our issue for July, 1914, page 431, we have been advised that the process has been improved by the addition of a refining method that yields high-grade silver bullion from the base precipitate of silver, copper and lead. This is a step that is expected to add to the profit of treating low-grade ore containing several metals, and we are glad to record the reported shipment of about 20,000 ounces of silver of 975 fineness. Formerly all of the valuable metals were precipitated as a base sludge and sold without refining, but silver is now separated.

Another chlorination mill in the Park City district, which is expected to be in operation in August, is that of the Park City Milling Company. This is a custom mill which will treat ore from the American Flag and other near-by mines. The process is similar to that used at the mill mentioned above, and the success attained at that mill is encouraging to the promoters of the custom plant. The latter was remodeled from an old dry concentrating mill where the Grasselli Chemical Company produced vinc concentrates from the tailings of wet concentration.

Dry Concentration and Flotation at Joplin

Coincident with the reported failure of dry concentration in the Joplin district comes the statement that the flotation process is to be given a thorough trial within the next few months. The mill of the Clifford Dry Concentrating Company was erected at Duenweg about a year ago, and was modeled along lines entirely foreign to ordinary local practice. Whether it was in the hands of incompetent operators or whether the process was wholly unsuited to the conditions, it is certain that the results were disastrous and little or no concentrates were ever produced. Flotation, however, is now in use in the Joplin district, and experimental work will be continued throughout the summer. Representatives of the state and national Bureau of Mines are said to be investigating the process, particularly in connection with a more perfect recovery of zinc.

The state of Missouri has made plans for the representation of its mining industry at the Panama-Pacific exposition in San Francisco next year. A sum of \$6,000 has been appropriated for making a suitable display

which will include model operations of drilling, mining and concentrating. The latter will be illustrated in a 25-ton plant. Lead and zinc smelting methods will be shown by means of models, photographs and finished products, and other operations related to mining and metallurgical industries will be displayed. The arrangement of the exhibit will be under the direction of Mr. Otto Ruhl.

Aurora Consolidated Sold

As a result of competitive bidding on the part of a number of mining companies the property of the Aurora Consolidated Mines Company, of Aurora, Nev., has passed into the control of George Wingfield, president of the Goldfield Consolidated Mines Company. The consideration is reported to have been about \$1,000,000. The Aurora property is one of the important mines of Nevada, and is located in a district that has a record of large production. In addition to the mine there is a 500-ton all-climbing cyanide mill which is now in operation. The ore is said to have a gross average value of about \$5 per ton in gold, and to yield a 92 per cent recovery by the process adopted. The vendors were Jesse Knight, of Provo, Utah, and his associates.

Consolidated of Carnotite Companies

As a result of the organization of the Consolidated Ores Company in Salt Lake City, nearly all of the carnotite deposits in the state of Utah have passed into the control of that company. The interests thus united are those of David Taylor & Co., the Unarium Mining Company, and the Moab Rare Metals Company. Over 125 claims of carnotite-bearing ground pass into the control of the new organization. Up to the present time all ore taken from these claims has been sold in Europe, but it is now the intention of the company to investigate at once the methods of treatment for low-grade ores, which continuing to market the high-grade material abroad.

Resumption of Smelting in Mexico

A matter of importance to the metallurgical industry is the reported resumption of smelting operations by the American Smelting & Refining Company at Chihuahua and other points in Mexico. It will be recalled that the company closed all of its plants in Mexico as a result of the revolutionary conditions and the consequent disturbance in transportation and other facilities for carrying on its business. Recently, however, the company has been ordering members of its technical staff to return to the smelting plants, and it is the intention to resume operations as rapidly as conditions will permit. The Chihuahua plant probably will be the first to be blown in, as it lies in northern territory and is less subject to disturbance. This will be followed by Monterey and Velardena, but the plant at Aguas Calientes may not be opened for some time.

Company Reports

The Buffalo Mines Co., Cobalt, Ontario, Canada, operates a concentrating mill for the treatment of low-grade silver ore. The sand tailing is stored awaiting the probable erection of a cyanide plant; the slime tailing is cyanided; and the concentrates from jigs and tables, and precipitate from cyanidation are treated by a special process of cyanide-amalgamation. From the report of the company for the annual period ended April 30, 1914, the following metallurgical data are taken. The average silver content of ore milled was 25.31 oz. per ton. The mill treated 77,616 tons and recovered 76.14 per cent by concentration. The cyanide annex handled 13,388 tons of slime tailing which contained an average of 10.13 oz. silver per ton, and made

a recovery of 77.8 per cent. The cyanide-amalgamation plant and refinery treated 20 tons of high-grade ore taken directly from the mine, 301 tons of jig concentrate, 923 tons of table concentrate, 6157 lb. of metallics from the mill and 17,347 lb. of cyanide precipitate. The total production of refined bullion was 1,682,759 fine ounces silver. The price received for silver during the year averaged about 59.5 cents per ounce. The cost of producing silver was 35.7 cents per ounce. The cost of concentration based on handling 77,616 tons of ore, was \$1.41. Cyaniding cost \$4.03 per ton of dry slime cyanided, or \$0.71 per ton of original ore.

The Iron and Steel Market

July has not altogether kept pace with June in exhibiting an improvement in the iron and steel market, but the improved conditions of June have practically continued through July, and this itself indicates an improvement in fundamental conditions, for July is normally a very dull month and a decided recession in activity would ordinarily be expected.

The first definite improvement in the buying of steel since the movement which began late in December and played out in February, began about June 1. For several months preceding shipments of steel exceeded the current orders, until the supply of old orders was practically exhausted and late in May steel production fell to a rate of somewhat less than 60 per cent of capacity. In June and July shipments have been at about 65 per cent of capacity, July having perhaps a slight advantage over June, while the bookings of orders for prompt shipment fully supported this rate of production.

Of contracting for forward delivery there was relatively little in June, while in July there has been a very fair movement, chiefly in contracts for third quarter. In the case of agricultural implement steel bar tonnage there has been general contracting through the year, and in many instances for the first half of 1915 as well. The total implement tonnage, however, has been relatively small, as much contract business was carried over July 1, contracts for the preceding twelvemonth having been very poorly specified. To a limited extent the automobile trade has contracted in sheets and other steel products through the early months of 1915. In the case of all the contract business extending into 1915, and in the case of a considerable part of that closed only for third quarter, advanced prices have been secured over the level prevailing for prompt shipment.

The steel mills in western Pennsylvania and Ohio, finding themselves in moderately comfortable position as to the volume of prompt business on hand, and with a considerable volume of contract business at advanced prices, announced an advance of \$1 a ton in bars, plates and shapes, about July 21, making the basis 1.15 cents, Pittsburgh. The Chicago district mills had been a trifle under the Pittsburgh basis, quoting an arbitrary of 15 cents per 100 lb. over the Pittsburgh basis instead of the full freight of 18 cents, and in that district there has been a slight tendency to advance prices proportionate to the Pittsburgh advance. The extreme Eastern mills have also shown a tendency to advance. More clearly than a month ago there is a zone market in steel products, the outlying districts not obtaining the full Pittsburgh price plus freight. The development of zone markets almost always occurs when demand is light and prices are now, the Pittsburgh basis being restored, as to practically all territory, when the market becomes active and strong. At this time, however, some doubt exists whether the Pittsburgh basis will ever come into full force again as to all products.

Effective July 20 the American Steel & Wire Com-

pany advanced wire products \$1 a ton, to a basis of \$1.55 for nails, independents promptly concurring. As usual, contracts had been booked with the large jobbers, providing for specifications to be filed within sixty days. The regular fall movement is about to begin, and if specifications upon old contracts are stimulated as expected, another advance will probably be made in the first half of August.

These advances represent the first sign of an upward tendency in steel prices since the beginning of February, and with a seasonable expansion in trade in August, coupled with good influences expected from the enormous crops in prospect the steel trade may be found not simply to have made a little turn for the better but to have entered upon a period of important improvement. Stocks are very low throughout the country and in such circumstances it usually requires only a definite start to be recorded in order to bring about a general buying movement.

Pig Iron

The pig iron buying movement which started about June 1 did not attain the proportions which would ordinarily be expected. June was no more than a fair month for buying, while July has been relatively dull. Prices have been almost stationary, except for a decline of 25 cents in Southern iron. The technical position of the pig iron market is fairly good, with prices so near the cost of production that there can hardly be any further important declines, while production and consumption are about balanced. Prospects naturally are for an increase in consumption rather than a decrease after such a long period of liquidation, and the idle furnaces, as a rule, will have to be tempted by higher pig iron prices before they will go into blast. Three or four merchant furnaces in the Central West, however, have under consideration the taking of chances for an advance, and getting into blast for the purpose of piling iron. The market is quotable as follows: No. 2 foundry, f.o.b. Birmingham, \$10 to \$10.25; delivered Philadelphia, \$14.25 to \$14.50; f.o.b. furnace, Buffalo, \$12.75 to \$13; f.o.b. furnace, Chicago, \$14; f.o.b. valley furnaces (90 cents higher delivered Pittsburgh) Bessemer, \$14; basic, malleable and No. 2 foundry, \$13; gray forge, \$12.50 to \$12.75.

Steel

There has been a great deal of price adjusting on contracts for billets and sheet bars and very little in actual transactions. The market definitely weakened early in July, with freer offerings by steel mills and at reduced prices, but at the close of the month a slightly firmer tone is reported. The market is quotable at \$19 for billets and \$19.50 for sheet bars, f.o.b. maker's mill, Youngstown, and at \$19.50 for billets and \$20 for sheet bars, f.o.b. maker's mill, Pittsburgh. Rods are about \$25, Pittsburgh.

Finished Steel

Bars, plates, shapes and wire products are \$1 a ton higher, as already indicated. Quotations are f.o.b. Pittsburgh, unless otherwise quoted, but in western and eastern territory the Pittsburgh basis is not fully observed.

Rails, standard sections, f.o.b. mill, except Colorado, 1.25c. for Bessemer, 1.34c. for open-hearth.

Plates, tank quality, 1.15c.

Shapes, 1.15c.

Steel bars and bands, 1.15c., base.

Iron bars, 1.25c., Pittsburgh; 1.17½c. to 1.20c., Philadelphia; 1.05c., Chicago.

Sheets, blue annealed, 10 gage, 1.35c.; black, 28 gage, 1.80c.; galvanized, 28 gage, 2.75c.; painted corrugated, 28 gage, 2.00c.; galvanized corrugated, 28 gage, 2.80c.

Steel pipe, ¾ in. to 3 in., 80 per cent off list.

Steel boiler tubes, 3½ in. to 4½ in., 72 per cent off list.

Wire nails, \$1.55; smooth fence wire, 1.35c.; galvanized barb wire, 1.95c.

Standard railroad spikes, 1.40c., Pittsburgh; 1.50c., Chicago.

Structural rivets, 1.55c.; boiler rivets, 1.65c.

Cold-rolled shafting, 67 to 68 per cent off list.

Chain, ¾-in. proof coil, 3.00c.

The Iron and Steel Institute

The annual meeting of this institution was held on the 7th and 8th instant in London under the presidency of Mr. Adolphe Greiner, whose address dealt mainly with the by-products of steel manufacture.

With regard to coke ovens, he said, they now worked for something else than the silvery metallic looking coke of the Durham beehive ovens. The resulting gas was formerly employed almost entirely for heating the ovens, but on account of its richness it was more profitably employed for metallurgical purposes and for lighting. It might in future be used for the production of nitric acid and, possibly, also for the hydrocarbons from which it was expected the synthesis of artificial rubber would be effected.

He advocated the removal of dust from blast-furnace gases, and eulogized the work of Mr. B. H. Thwaite, who demonstrated the advantage of the direct use of those gases for the production of electrical energy. There was a considerable number of metallurgical works running between five and ten blowing engines and from ten to fifteen generators, some of them showing an aggregate output of over 30,000 hp, for which coke-oven gas, blast-furnace gas, or both, were utilized. Valuable economy, to the extent of 10 to 15 per cent results from using the heat of the exhaust gases for steam raising.

Besides the gases there were also slags which were already being used for making a kind of Portland cement, and also for bricks.

There was still much to be done in modern steel works in effecting economies by utilizing heat which was now lost. Engineers might some day solve the problem of avoiding the tremendous heat losses now incurred in the converters; and heat losses by radiation from reheating furnaces and so on would repay their serious attention. Again many rolling mill steam engines—particularly in reversible mills—discharged into the atmosphere large volumes of steam which might be profitably utilized in low-pressure turbines.

If metallurgical works produced in regenerative ovens the whole of the coke required for the blast furnaces, and if the slag were used for making bricks and cement, and the waste gases were used for the production of motive power, he estimated that the resulting economy would amount to about 9s. 6d. per ton of pig iron, 11s. 2d. per ton of steel ingots, and 12s. 6d. per ton of rolled steel, and that without taking into consideration the sum yielded by the sale of basic slag.

Montreal Meeting of American Chemical Society.

The date of the Montreal meeting has been changed from Sept. 8-11 to Sept. 15-18, 1914 (Tuesday to Friday).

Johnson Electric Zinc Plant.—Johnson Electric Smelting, Inc., has contracted for power with the Mississippi River Power Company for a plant at Keokuk, Iowa, where the Johnson electric furnace for the smelting of complex zinc ores will be installed in commercial operation.

Leipzig Meeting of the German Bunsen Society

The twenty-first general meeting of the German Bunsen Society for Applied Physical Chemistry was held in Leipzig from May 21 to 24. It was called to order in the morning of May 22 in the lecture hall of the International Exposition on Books and Graphics by the president of the Bunsen Society, Prof. M. LeBlanc. Speeches of welcome were made by Geheimrat Schmalz, of the Department of Education of Saxony; by Geheimrat O. Mayer, president of the University of Leipzig; by Dr. Dittrich, mayor of the city of Leipzig; by Dr. Volkmann, chairman of the board of directors of the International Exposition on Books and Graphics; by Geheimrat Weinstein, representing the Reichsanstalt and the Bureau for Testing Materials, and by Dr. Krey, who spoke for the German Chemical Society and the Association of German Electrical Engineers.

The Bunsen Medal was conferred on Prof. Walter Nernst, of Berlin, who being absent on a lecture tour in South America, had sent a paper on the application of his new thermodynamic principle to gases. This paper was read by Prof. von Warthenberg.

In the business meeting held on May 23 in the lecture hall of the Institute for Physical Chemistry of the University the new officers of the society were elected.

Prof. Hans Goldschmidt, of Essen-Ruhr, Germany, the distinguished inventor of the aluminothermic method, is the new president, his term running from July 1, 1914, to June 30, 1917. For the same period Prof. M. LeBlanc, of Leipzig, was elected vice-president. Geheimrat Dr. von Boettinger was re-elected treasurer.

Professor Goldschmidt, after expressing his thanks to the society, referred to the preponderance of scientists in the Bunsen Society. This is due to the difficulty which engineers have in presenting papers before the society. On one side they are exceedingly limited by commercial considerations in the extent to which they can discuss their work. On the other hand, they must avoid the appearance of speaking "pro domo," which means in this case "for their pocketbook." Nevertheless the presence of engineers in the meetings of the Bunsen Society and their mingling with the scientists is highly important; they can give and take. In recent years in both scientific and engineering circles in Germany there is felt a certain lack of able young assistants. In parenthesis Dr. Goldschmidt remarked that graduates of modern German woman universities have not the slightest difficulty in finding good positions in science as well as engineering, not for their looks but for their ability. But the demand for young able chemical assistants is now greater in Germany than the supply, and if this situation should continue it would mean a national calamity. In conclusion Dr. Goldschmidt referred to the work of Bunsen, who was a scientist but always sympathized with the work of the engineer. As many of his students have become engineers as scientists. The influence of the Bunsen Society should manifest itself in the same direction.

According to the report of the board of directors the number of members on April 1, 1914, was 792. The assets of the society amount to over \$8,000.

The numerous papers presented at the meeting will all be published in full in the organ of the Bunsen Society, the *Zeitschrift für Elektrochemie*. Abstracts of the papers appeared in the issue of June 1, while publication of the papers in full has begun in the issues of June 15 and July 1 and is to be continued.

Dr. P. Klemm spoke on the application of the principles of physical chemistry to paper manufacturers, Dr. R. Rübenkamp on printing colors, and Prof. E. Goldberg on engravings and special presses for engravings.

Electroplating

An interesting review of the present status of electroplating was given by Dr. W. Pfanhauser, of Leipzig. He estimated that purely for electroplating work in Europe alone, 2,500,000 amperes are in use, and in the whole world 3,000,000 amperes. This corresponds to 15,000 tons metal deposited per year in electroplating (not including electrolytic refining). Thirty-five per cent of this work is done in Germany and Austria.

Besides nickel plating, zinc plating is now coming to the front. For zinc plating of wires, ribbons, and tubes, 300,000 amperes are now in use. The main interest of progressive electroplaters is at present directed toward improvements in mechanical appliances. But fundamental changes in the chemistry of electroplating may also come and the author thinks that the next chemical revolution will be the use of fused electrolytes.

Indefiniteness of Atomic Weights

Two rather revolutionary scientific papers, both coming to the conclusion that the atomic weight of an element has not a definite

value were presented by Dr. K. Fajans, of Karlsruhe, and by Prof. O. Hönigschmid, of Prag, respectively.

In his paper "on the final products of radio-active disintegration series" Dr. K. Fajans said that until recently it has been assumed that a chemical element has always the same atomic weight from whatever mineral it may have been isolated. But when a year ago the author and Soddy brought the radio-active elements in line with the periodic system, the hypothesis was suggested that there are elements which while entirely identical in chemical respects may differ in atomic weight up to eight units and which differ otherwise only by their radioactive properties (rays, speed of disintegration). This hypothesis was first based only on indirectly determined properties of radioactive elements of short life and it was therefore important to prove experimentally that two elements which appear to us to be identical chemically can have different atomic weights.

The author then concluded that this test could be made with lead. He considered it as probable that the atomic weight of lead formed in the course of millions of years in uranium minerals as the final product of the disintegration of uranium would be different from that of lead obtained from ordinary lead minerals.



HANS GOLDSCHMIDT,
PRESIDENT GERMAN BUNSEN SOCIETY.

This conclusion has been tested and has been fully confirmed by the author's former collaborator, M. Lemberg, in the laboratory of Prof. Theo. W. Richards in Harvard.* Lead from uranium minerals was found to have the atomic weight 206.6, while ordinary lead has the atomic weight 207.1. The possible error is at most one or two units in the last decimal and is therefore much smaller than the difference found experimentally.

Of greatest interest is the fact that these two kinds of lead are not only chemically identical, but also identical in spectroscopic respect. On the basis of similar considerations it is probable that lead from thorium minerals has a higher atomic weight than ordinary lead. Possibly ordinary lead is but a mixture of lead from uranium and lead from thorium. The question comes up whether the atomic weights of most elements are perhaps only mean values of the atomic weights of several chemically identical elements which therefore cannot be separated by chemical means. Atomic weight researches have therefore to meet an entirely new problem; that is to investigate how the atomic weights of the elements depend on their origin.

The paper by Prof. O. Hönigschmid on a "revision of the atomic weight of uranium" gives the results of a determination by analysis of uranium bromide. His atomic weight of uranium is 238.18, that is 0.3 units below the present international value. He considers the value 226.4 for the atomic weight of radium to be improbable, since the difference between the atomic weights of uranium and radium must be at least 12, that is, 3×4 (atomic weight of helium). His own value $Ra = 225.97$ is therefore considered to be more probably correct.

At the same time the author determined, jointly with Stefanie Horovitz, the atomic weight of lead obtained from uranium ore and found 206.73, that is, 0.4 less than the atomic weight of lead obtained from ordinary lead ore free from uranium.

Other Papers

Prof. E. Cohen, of Utrecht, spoke of the metastable state of metals. By dilatometric methods it has been found that the metals as we know them now are metastable and contain simultaneously several allotropic modifications. For this reason the metals are not in equilibrium at ordinary temperatures and gradually disintegrate. The author reported on new experiments in which he obtained by very sensitive electric methods the same results as by dilatometric measurements.

Prof. H. von Wartenberg, of Danzig, discussed the possibility of the existence of gaseous alloys; Prof. F. Fichter, of Basel, spoke on the Kolbe reaction with $\text{HOOC}-\text{CH}_2\text{SO}_3\text{H}$; Prof. H. Goldschmidt, of Christiania, on the equilibrium between hydrogen ions, alcohol and water; Prof. A. Hantzsch, of Leipzig, on indicators; Profs. F. Haber and G. Just, of Berlin, on new experiments on the emission of electrons in chemical reactions; Prof. G. Bredig, of Karlsruhe, on the chemical kinetics of formic acid formation; Prof. H. Bechhold, of Frankfurt on Main, on the existence of colloidal solutions of sodium urate; Prof. R. Marc on the kinetics of adsorption; Prof. M. LeBlanc on the photobromination of toluol; Mr. N. S. Hatfield on some new electric apparatus (a hydrogen-electrode meter; a flame arc lamp with improved efficiency and diminished carbon consumption; a time switch without mechanism); Prof. O. Sackur on the equation of state of gases and the theory of quanta; Prof. W. Loeb on some fundamental chemical problems on the effect of high-

tension electric currents on the growth of plants; Dr. G. Schulze, of Charlottenburg, on electrolytic valves; Prof. W. A. Roth, of Greifswald, on graphite, diamond and amorphous carbon; Dr. V. Falcke, of Leipzig, on the reaction between ferrous oxide and carbon and between ferric oxide and iron; Dr. A. Goldmann, of Leipzig, on the theory of the Becquerel effect; Dr. K. Schaefer, of Leipzig, on optical investigations of the constitution of inorganic acids, salts and ester; Dr. M. Volmer, of Leipzig, on photochemical sensitiveness and photo-electric conductivity; Dr. E. Wilke, of Heidelberg, on the Tyndall phenomenon, and Dr. M. Trantz, of Heidelberg, on two gas reactions.

Next year's meeting will be held in Karlsruhe, beginning May 13, 1915.

The Non-Ferrous Metal Market

Beginning July 1 the non-ferrous metal market gave evidence of greater activity and firmer prices. This followed the marked depression during the latter part of June when prices fell off materially and sales were small. Copper and tin have benefited most by the change, while lead and spelter continue to remain quiet and fairly inactive.

Copper.—Prices for this metal advanced from the low figure of 13.20 @ 13.25 cents in the latter part of June to about 13.55 @ 13.65 early in July. The market stiffened and sales abroad were greatly increased, while local consumers also bought in larger quantities. Nevertheless stocks have increased, as shown by the report of the Copper Producers' Association for June.

Tin.—Favorable statistics on this metal published on the first of this month affected the market and added to the general tendency toward higher prices than prevailed in June. July tin is quoted at 32 cents in the domestic market.

Lead.—The New York market has remained firm but the Western market is somewhat lower in price and quiet. New York quotation is 3.90 cents, and St. Louis 3.72½ @ 3.75 cents.

Spelter.—Prices for this metal improved slightly from those of June, but the market is quiet and without feature. New York prices are 4.90 @ 4.95 cents, and St. Louis 4.75 @ 4.80.

Other Metals.—The market for aluminium is poor and sales are small. The metal is quoted at 17½ @ 18 cents per lb., New York. Business in antimony also is quiet, but prices are firm, ranging from 5.65 to 7.35 cents for various brands. Quicksilver is quoted at \$37.50 @ to \$38.50 for flasks of 75 lb., New York, and \$37.50 at San Francisco.

Specific gravity separation applied to the analysis of explosives is treated in **Technical Paper 78** of the Bureau of Mines. The authors are C. G. Storm and A. L. Hyde.

The Richards Pulsator Riffle is an application of the Richards pulsator jib to riffles such as are used in hydraulic mining operations or on dredges for the recovery of gold. The ordinary Hungarian riffle consists of angle irons placed across a sluice or table at right angles to the flow of water and gravel. A section across the riffle appears like a series of inverted letters L. In cleaning up the accumulation of gold from the riffles, the angle irons are removed and the concentrate taken out by hand. By applying a pulsating current of water under these riffles, according to the principle of the Richards pulsator jig, an automatic riffle with continuous discharge is produced. This device has been tested by the manufacturers, the Denver Engineering Works Co., and found to give excellent results on placer sand and gravel.

*See also the paper by Theodore W. Richards and Max E. Lemberg on "The Atomic Weight of Lead of Radioactive Origin," since published in the July issue of the *Journal of the American Chemical Society* (vol. 36, p. 1329).—Editor.

The Construction of the Blast Furnace Stack*

BY J. E. JOHNSON, JR.

(Concluded from page 460)

The Shaft or Inwall

Above the mantle only two styles of construction have been used in recent years. For many years nothing but plain fire brick from two to five feet in thickness was used from this point to the top of the furnace, but the tendency for the lining to be eaten away more extensively just above the top of the bosh than anywhere else, which had existed very generally for many years became worse with the general use of fine ores, and the lining troubles which they brought with them.

As the cooling plate construction had given a good account of itself in the bosh, furnace-men began to put rows of cooling plates above the mantles to prevent this action, and this practice is now quite generally followed. These were at first simply built in the brick work and the pipes to them were brought in through small holes cut in the shell for that purpose. This, of course, put these cooling plates entirely beyond the possibility of renewal, if they failed they were done with and must be shut off and were soon burnt up.

To meet this situation some furnace men have cut holes through the furnace shell large enough to permit the removal of the whole cooling plate exactly as is done on the bosh, but this cuts up the shell very badly and even when this provision is made it is a matter of much difficulty to remove these plates through such a great thickness of brick work. It is, therefore, doubtful whether the trouble and expense of making these plates removable is worth the small gain to be obtained by that means. It will be seen that in the case of the furnace shown by Fig. 4* the course was adopted of bringing the water pipes for these blocks out through the shell so as to give access to the several connections but without making any provision for the removal of the cooling plates.

In early days of cooling plates they were made of cast iron with coils of pipe cast in them, these in all cases being built solid in the brickwork, without any provision for removal. The pipes were first made in one continuous coil zigzagging back and forth through the cooling plate, but an improvement on this was subsequently introduced which consists in making each pass of the pipe through the block separate and bringing the ends of all the pipes out through the outside surface of the plate, and putting the connections which put them in series on these projecting ends. By this means if the inside pipe failed it could be cut out and the one back of it left in service. Similarly if this failed by the gradual burning back of the furnace it in turn could be cut out and so on back for as many passes as the pipe might make through the plate, this number generally being from three to five. This arrangement is shown in Fig. 7.

For the protection of the bosh, hollow bronze cooling plates which do not adhere very tightly to the brickwork and on the inner surface of which the iron and slag do not burn fast in great lumps, so as to prevent their removal, were a great improvement, because they could be replaced as often as they failed and inside of the furnace kept practically to the original lines for a long period of time.

On account of this superiority for service below the mantle, the bronze cooling plate has often been used above the mantle also, but after considerable experi-

ence with it I prefer the cast iron plate with pipes cast in, for that service, for two reasons.

First, these plates above the mantle being the highest of any of the water-cooled portions of the furnace they are the first to lose the water in case of any partial failure of the water supply which reduces its head. I have known of several cases where this happened with-

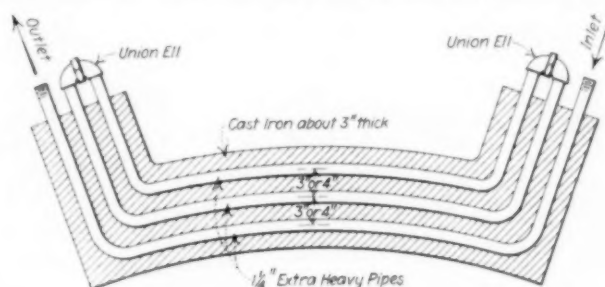


FIG. 7—CAST-IRON COOLING PLATE

out the complete failure of the water supply. This means that these cooling plates above the mantle will lose their water first, and if they do so the bronze plates are bad for two reasons. First the melting point of the bronze is many hundred degrees below that of the cast iron plate and its mass is much smaller, the body of the plate being hollow, so that when the water on these plates has been lost, even though the furnace be stopped at once, the heat of the surrounding materials will cause the bronze to melt after a comparatively short period, which leaves a considerable cavity in the brick work into which the bricks from above are likely to settle, thus causing at least the possibility of a crack in the lining, even though the gas does not blow out through the burnt cooling plate itself.

With the cast iron plate, on the other hand, the conditions are quite different. The melting point of cast iron is not reached by the contents of the furnace at a point much above the mantle, so that these plates are much less likely to be melted out completely than the bronze plates, and even if the inner end next the stock is melted off the outer end is protected by the brickwork for a long time, if not indefinitely, and being practically solid except for the pipes cast through it a partial melting of the cooling plate does not form a cavity as in the case of the bronze plate.

Secondly, if even a small portion of the nose of the bronze plate be melted off it is destroyed completely, as this opens the water space into the furnace, but with the multiple pipe system several inches of the plate may be melted away and still, by using some of the outer passes of pipe, water may be put through it again, and its usefulness as a cooling member preserved indefinitely.

When these plates are installed, whatever kind may be used, provision must be made for obtaining access to them. In Fig. 4 two platforms for this purpose will be seen above the platform at the level of the mantle, these platforms being provided with a suitable handrail. This is important because in case of trouble with the circulation in these plates it is necessary to obtain access to them quickly in order to prevent their being burnt, and this cannot be done if ladders have to be arranged for the purpose. Moreover there is always a certain quantity of furnace gas stealing up the walls of the furnace, no matter how well it may be constructed, and while this may be so dilute that the workmen exposed to it will not realize the danger, he may nevertheless be made unconscious by it. Under such conditions, if working from a ladder he is almost certain to fall and be seriously injured or killed, whereas if a secure platform with a good handrail be provided

*Figures 1 to 6 were published in the instalment of the July issue.

he will fall upon this and can easily be rescued before further poisoning by the gas.

The number of rows of cooling plates above the mantle is a matter of the individual conditions and the judgment of the operator. The customary number is probably from three to six rows, although in several cases these rows have been run all the way to the top of the furnace, spaced about as shown in Fig. 4. This is often done with furnaces intended to be run on ferro manganese or spiegel which are highly destructive of their linings. For ordinary furnace practice the consensus of opinion is that there is no advantage in having more than a limited number of rows of these plates.

There is a drawback to the use of these plates in that while they prevent the lines of the furnace from cutting back at the plate they do not prevent heavy erosion in the space between the adjacent rows very much as shown for the bosh in dotted lines in Fig. 3 and the steps so formed offer a lodgment for scaffolds in the very region of the furnace where the stock is in its pastiest condition and therefore most likely to scaffold. Consequently these steps tend to promote scaffolds with all their disastrous consequences.

For this reason some operators prefer not to use cooling plates above the mantle at all, but to let the lining wear away and cool the shell externally with sprays if it becomes necessary. It is stated by Mr. H. A. Brassert in his paper on Modern American Blast Furnace Practice read before the American Iron and Steel Institute, May, 1914, that "cooling plates above the mantles have only survived where they are placed from 18 inches to 22 inches back of the face of the inwall and in this position they have not much effect in preserving the original lines."

This statement may not be universally true, but may be considered as absolute proof that cooling plates above the mantle are not a panacea for lining troubles but on the other hand are only to be used sparingly and with great discretion.

From this region at the base of the shaft to a point a few feet below the stock line conditions of the temperatures and abrasion are such that well made fire brick alone will withstand a long campaign, and consequently in the great bulk of modern practice nothing else is used.

The Stock Line and the Top of the Furnace

In this region the influences of chemical activity and temperature are greatly reduced over what they are in the lower portions of the furnace, but the lining is exposed to another form of attack which has proven almost as difficult to resist as that in the hearth and bosh.

This is the abrasive action of the stock as it is dumped from the conical bell into the furnace. The charge of coke in a modern furnace is from six to twelve thousand pounds, and in the best modern practice this is an extremely hard dense coke. In practically all cases also the limestone is in the form of lumps; sometimes this is crushed to go through a three or four-inch ring, but in many cases lumps which would scarcely pass through an eight-inch ring are tolerated, such lumps weighing up to twenty pounds.

When the bell opens these slide off its steeply inclined surface and strike violently against the walls of the lining at the stock line. In some cases also a considerable portion of the ore consists of hard lumps, some forms of ore having not only a high specific gravity, but also a degree of hardness comparable to that of granite, though these conditions are not usual in most present practice. These materials with their hard and abrasive nature inpringing on the stock line at a rate varying from one to two thousand tons per

day cannot fail to exercise a destructive effect upon any brick work used for the lining, and as a consequence many forms of metallic protection have been used for this portion of the furnace.

The earliest form of protection known to me was angles of cast iron, six inches square on the face, six by nine inches in the body, by one inch thick. These were laid in fire clay on top of a course of brick work, and another course of brick work six inches thick built on top of them, then another row of angles was inserted, and so on. This armour extended for some eight or nine feet vertically in the furnace from a point two or three feet above the stock line to one seven or eight feet below it.

These after many years' trial proved to be unsatisfactory. The ends of the plates would break off and the protection of that area would be lost, but more serious still, cast iron swells when repeatedly heated and cooled through certain temperature ranges, and these are ranges which exist at the top of many furnaces.

It is probable, moreover, that a graphitization of the carbon in these plates occurs in consequence of their long heating, and this also is accompanied by swelling which tends to disintegrate the structure of the angle. Moreover, this swelling took place unequally at different points around the circumference of the furnace and the top of the brickwork was not only pushed up to an objectionable extent by the action but also at some points more than at others.

At other times and plants steel mantles or circular shields slightly smaller than the brickwork were hung inside the furnace at the stock line, but these failed because while the temperature at that point is not normally high enough to affect the structure of wrought iron or steel, it is high enough to cause those materials to warp very extensively in course of time, and if these mantles warped at all they must go inward since the brickwork close outside them prevented them from warping outwards. Such irregular projections of portions of the mantle into the furnace utterly destroyed the regularity of the stock line upon which the proper filling of the furnace depends, and brought about the very conditions which the mantle was introduced to avoid.

Cast iron or steel rings built up of segments were tried at some plants. These were built into brick work practically like a course of iron or steel bricks at intervals of two or three feet vertically, but as the stock has a considerable outward velocity as it leaves the bell these did not prevent it from cutting into the brickwork between them, and they, therefore, failed to fulfil the purpose for which they were intended.

I have never used these myself and I do not know of any case where they have been continuously used by any one plant for a long series of years.

One of the earliest successful means of solving this troublesome problem was an adaptation of the old cast-iron angles, first made by the writer so far as known to him, in 1903 or 1904, at the plant of the Longdale Iron Company.

After realizing the uselessness of the cast-iron angles, the suspended mantles, and various other schemes, it seemed likely that wrought iron being much lower in carbon than the cast iron, would be free from the tendency to grow by repeated heating and cooling, and would have also the physical strength needed to prevent breaking off the vertical portions of the plates in service. It would also be free from the liability to extensive warpage which had destroyed the steel shields because the individual angles, six inches square on the face, would have between them spaces of an eighth of an inch or so at the closest, and these spaces pro-

viding room for individual expansion would prevent objectionable warpage which was the result of the cumulative expansion of a considerable area of the mantle as a whole.

Accordingly a trial of such angles was made. These were made of three-quarter-inch by six-inch wrought iron, six inches square on the face, six inches by nine on the portion built into the brick work, and were successful far beyond our highest expectations. Instead of being compelled to reline the top of the furnace at the end of every campaign as had previously been the case, a furnace would make two or three campaigns without disturbing the upper portion of the lining.

Since that time these angles have been used with some modifications at a considerable number of plants, and always, to the best of my knowledge, with entire satisfaction and success. Modifications have been made in some cases. The portion of the angles built into the brickwork has sometimes been made long enough to go entirely through the brickwork, and the outer end turned up into a hook, this being imbedded in concrete poured around the lining at the top of the furnace. I have never seen any tendency for these angles to work out of the brickwork which would necessitate this additional construction.

There is, however, a very simple method of preventing this, if such danger were anticipated by a simpler and less expensive means. The leg of the angles built into the brickwork can be sheared to approximately the taper which will make them lie to the circle of the furnace. If the sides of the plates before bending were sheared to an unbroken straight line this would result in slight gaps between the turned-down portions or faces of the angles, but this would be a matter of no importance and the portion of the angle built into the brickwork, having its largest end out, would be absolutely prevented from working forward into the furnace.

It is something of a nuisance to level up the outside courses of the brickwork so as to bring them level with the tops of these angles, when the latter do not run clear through the wall, and to avoid this difficulty bricks of different thickness may be used on the inner row where the angles are laid. By using what firebrick makers call the standard thickness of brick in the courses on which the angles are to come, and the three-inch thickness in the rear courses behind them, the thickness of a three-quarter-inch angle is just about compensated with a minimum of difficulty in levelling up and without using any but standard material.

There are no patents on the use of these angles made of wrought iron or mild steel though a patent on their use when made of hard steel is held by Mr. E. B. Cook.

Another type of metal protection for the top of the furnace is one designed by Messrs. Albert Broden, manager of the Reading Iron Company, and Frank C. Roberts of Philadelphia. This is shown by Fig. 8 and consists of a cast steel sleeve the shape and size of the furnace itself at the stock line, which rests on cast steel brackets securely riveted to the shell.

The downcomer opening is cast through this sleeve and all brickwork is left out between it and the shell of the furnace proper. Large openings are cut through the shell to render the sleeve accessible for expansion and to provide air cooling for it. Water troughs are cast in its outer surface and provision made for a spray pipe around the top so that a continual trickle of water may be kept up over its surface in case of necessity.

The bottom section is cast separately and is cooled by water pipes cast in. This projects down inside the top of the brick work and protects the upper edge of the latter from the abrasion of the stock. The cooling water is trapped from the pockets in the back of the

brackets, which are fastened water-tight to the shell and so prevent any possibility of water getting down through the brickwork. The joint with the hopper at the top is made by a single layer of brick laid in the usual way. This is an excellent construction and has given good results since it was put into service a short time ago. The only objection to it is the expense, being made of steel castings with accurately machined joints at all points where the different portions come together, the construction is a very expensive one.

The Shape of the Top of the Furnace

It was formerly customary to run the shell practically vertical to the top, putting brackets in the inside to support the hopper ring and other brackets on the outside to carry the wheelway for the filling barrows.

In early days the hopper ring was not even bolted

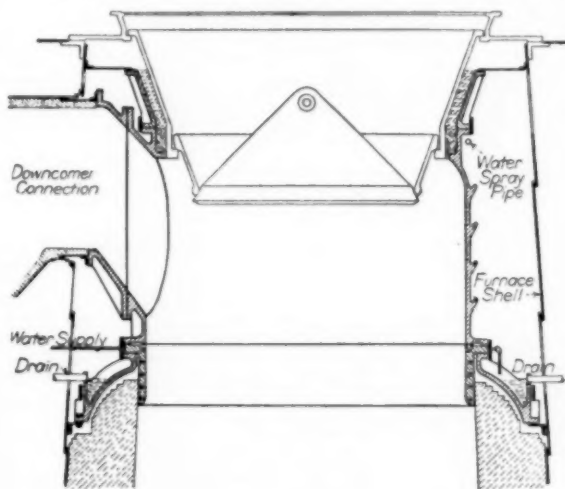


FIG. 8—BRODEN-ROBERTS STOCK LINE PROTECTION

down to the brackets but with the greater frequency of violent slips which followed immediately upon the introduction of large percentages of fine ore into the burden, it was soon found necessary to have a more substantial construction. This was begun by making the brackets of steel instead of cast iron and bolting the hopper and hopper ring to them. Finally Mr. Julian Kennedy pointed out the general weakness of this construction and the necessity of holding the top of the furnace down until the violence of the slip had passed by. In fact he claimed that by preventing the release of pressure at the top the violence of the slips could be much diminished, and he was the first to introduce the coned top shown in Fig. 2 in which the shell is drawn in to support the hopper ring directly, thus making a much better and tighter joint between the hopper and the shell and giving a construction of sufficient strength to resist all ordinary slips and explosions.

The wheelway of the hand filled furnace is of course entirely eliminated as such, in modern mechanically filled furnaces, but an operating platform is provided, generally about as shown in Fig. 2.

The great mass of brickwork formerly put in above the bottom of the hopper is in fact perfectly worthless as it has no heat-resisting functions to perform and only serves to make a joint between the hopper and the furnace shell, and generally a very poor joint it was, having to be supplemented by clay packing, and accompanied by more or less leakage of gas, particularly during the blowing-in periods of the furnace when the gas is most deadly. The elimination of this construction has also resulted in putting the gas outlets further up toward the top of the furnace where they

belong, somewhat as shown in Fig. 9 which is an elevation of the furnace shown in Fig. 2 and shows the location of the gas outlets more clearly.

Thin-Lined Furnaces

It has been recognized for many years that fire brick exposed to the conditions in the blast furnace would wear back until the amount of heat carried off by their bases about equalled the amount brought in by their inside surface, after which time wear took place much more slowly. It has also been recognized

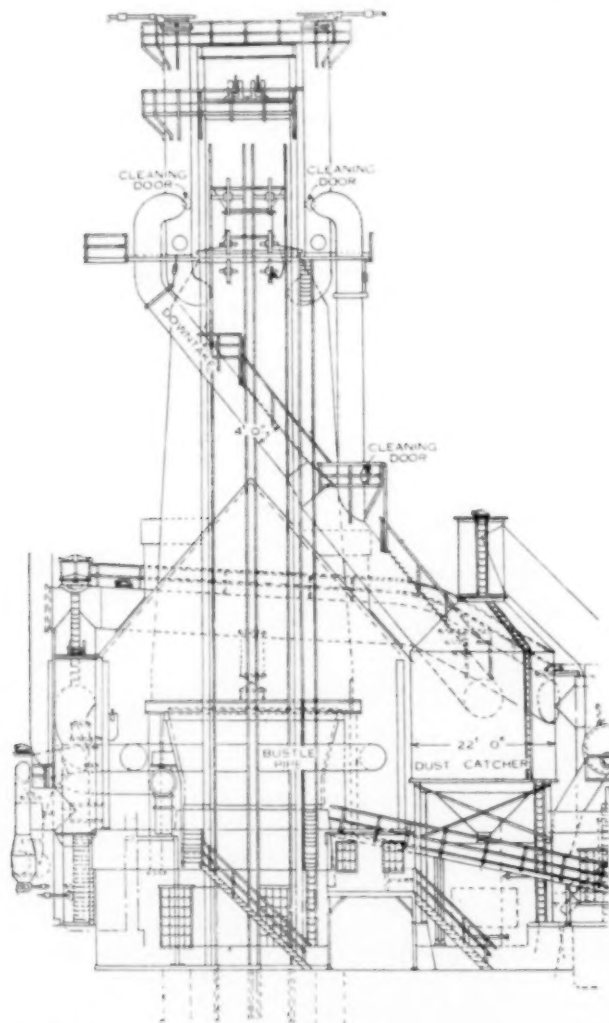


FIG. 9—ELEVATION OF BETHLEHEM STEEL COMPANY'S "A" FURNACE, SHOWING DOWN COMER

that in the standard construction considerable life of lining can only be secured by a corresponding thickness, and that as this thickness wore away the departure of the furnace from its original and presumably correct lines became greater and greater, generally with more and more unsatisfactory working.

About fifteen years ago the experiment was tried in Germany with the idea of overcoming these difficulties of making the lining of the furnace exceedingly thin, only a few inches in fact, and cooling its exterior surface so as to maintain its original shape indefinitely. The idea has superficially much to recommend it, and after several years' trial in Germany it was introduced into this country, first at the Lucy Furnaces in Pittsburgh, under the management of Mr. James Scott.

The German construction consisted of cast-iron staves with flanges on their exterior surface and jaws like flanges on the inside, into which the bricks were

securely fastened. These were built up into a furnace shaft very much in the same way that the bosh jacket shown in Fig. 5 is built up, but the exterior was cooled by a water film or spray instead of the pipes cast in shown in Fig. 5.

The construction adopted at the Lucy Furnaces was different. There a steel shell was built much smaller in size in proportion to the lines of the furnace than the ordinary shell, since the thickness of the brick with which it was lined was only about one foot from top to bottom. On the outside of this shell were riveted annular troughs, each about two feet deep by six or eight inches radial width, and arranged so that water pumped into the top one flowed down through all the others in sequence to the bottom. In spite of great pains to secure absolutely air-tight construction, some water leaked into the furnace through the rivet holes and the results were unsatisfactory, though subsequently these difficulties were overcome.

Soon after this two thin-lined furnaces of this type were built at the Illinois Steel Works, and subsequently one of these was rebuilt with cast-steel plates instead of the boiler-plate shell and riveted buckets. This, however, did not prove satisfactory and the latter construction was restored. A bucket type of thin-lined furnace was also constructed at the plant of the American Steel & Wire Company in Cleveland.

A thin-lined furnace was subsequently built by the Detroit Iron and Steel Company according to the plans of their general manager, Mr. Max McMurray. This solved the problem of saving the original shell for supporting the top works of the furnace, which had not been done in any of the other constructions, columns having been run all the way up from the mantle to the top for this purpose.

In the case of the Detroit furnace the expedient was adopted of building a smaller shell with a thin lining inside the original shell and cutting a number of doors through to give access to the inner shell for examination, this shell being cooled directly by water sprays at the top forming a film of water which runs all the way to the bottom. The outside shell also presents the advantage of preventing the wind from blowing the water off the stack, thus leaving it dry on one side, and making a disagreeable spatter below, both very important points. This construction is shown by Fig. 10.

The Tennessee Coal & Iron Company have also had a thin-lined furnace in operation for a number of years. This was a very heavy boiler-plate shell with spray pipes arranged around the top. A series of galleries were arranged at different levels around this shell and outside these galleries was a sort of housing of steel slats to prevent spatter and to prevent the wind from blowing the water off the shell.

A different construction still has been used by Mr. Frank C. Roberts in a furnace built by him for the Warwick Iron Company, shown in vertical section in Fig. 11. In this the top of the furnace is supported by separate columns running up from the mantle, as previously described, and the shell is a boiler-plate shell, but instead of trusting somewhat to chance in the matter of the distribution of the cooling water around it, this is accomplished by a helical trough shown in larger section in Fig. 12.

The trough is supported by tap bolts which do not go all the way through the shell and, therefore, make leakage through it impossible. The bottom edge of the trough is spaced away from the shell by distance blocks as shown. Part of the water runs out the space at the bottom of the trough and that which cannot get out this way runs around the helical trough. In this way a perfect distribution of water is secured around

the circumference of the shell and the danger of wind blowing the shell dry on one side is eliminated.

A compromise, one-half thin-lined and one-half standard construction has been in use for several years at the Port Henry Furnace of the Northern Iron Company. This construction shown in Fig. 13 is that of the former manager of the company, Mr. F. E. Back-

ent from what they are below that point, because as already pointed out the convergent slope of the bosh provides a natural lodgment for the slag which runs down and chills upon its cooled surface in the normal operation of the furnace, so much so, that as before stated, if a furnace were started with an externally cooled bosh jacket entirely bare of bricks this would soon cover itself with a lining as good or better than

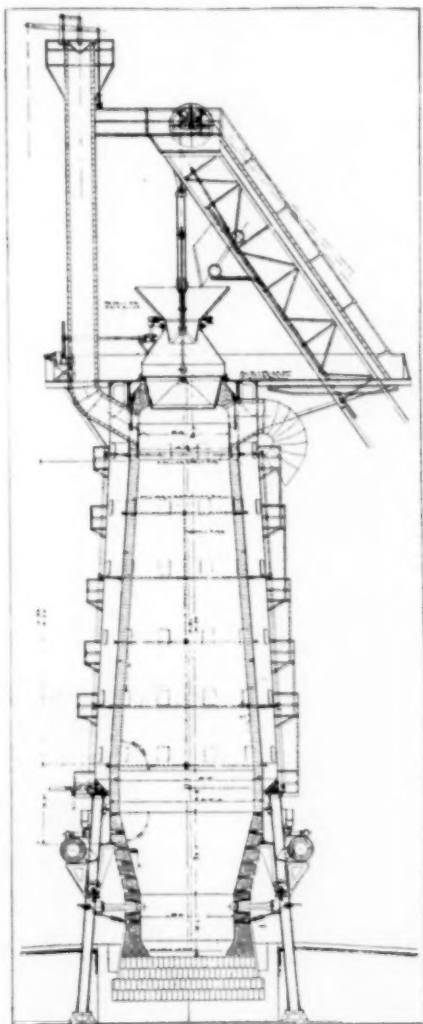


FIG. 10—DETROIT IRON & STEEL COMPANY'S THIN-LINED FURNACE

man, the design being carried out by Frank C. Roberts & Co. This furnace has a complete mantle shown in section resting on the vertical portion of the column at the level of the top of the bustle pipe. From this other columns run up on a slight inward batter to a second mantle some twenty-five feet higher. On this second mantle rests the upper portion of the inwall which is of standard construction. From this point down to the bosh there is a boiler-plate shell with rings of angles for supporting the brickwork riveted to its inside surface at intervals of five or six feet. This shell makes a joint on top of the cast-iron water-cooled segmental bosh jacket. The bosh jacket is lined with nine inches of brick and the nearly vertical portion of the shaft above it with thirteen inches.

This construction is intended to prevent the gouging out of the lining above the top of the bosh which has been described above. This furnace has now been in operation for several years and is said to have given good satisfaction.

The conditions in regard to thin lining for the inwall or any point above the top of the bosh are vastly differ-

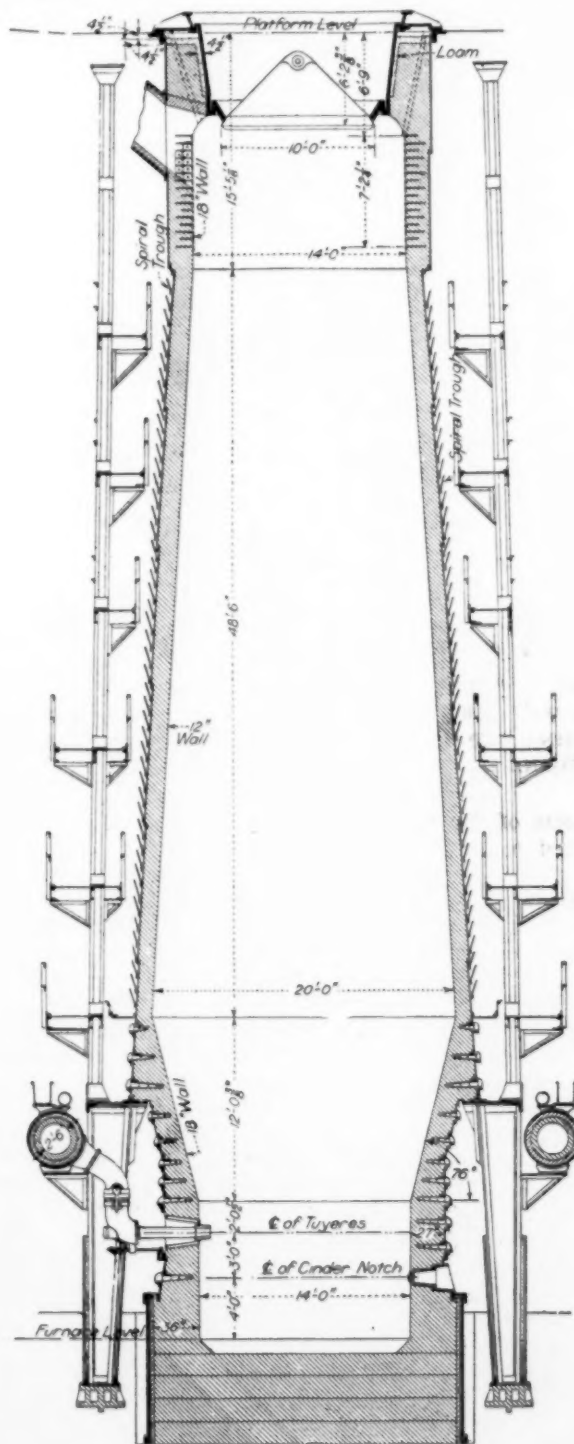


FIG. 11—SECTION OF WARWICK THIN-LINED FURNACE

the best fire brick which could be put upon it. But above the mantle the slope is in the other direction, the shaft of the furnace is a cone divergent downward, moreover except in its extreme lower portion there is no molten material, and if there were any it could not impinge upon this outward flaring surface, and could

not remain there if it did so. Therefore if the lining is once removed from this portion of the furnace it is gone for good and all, until restored by relining.

Moreover the furnace shell does not furnish much support in the vertical direction for any lining which may stand against it. It does prevent this lining from pushing out or falling backward, but it does not prevent its falling inward or sliding straight down in case of the failure of its support below. For this reason it seems to me necessary that the lining of the furnace above the bosh should consist of a sufficient thickness of brickwork to support itself and the portion of the lining above it. Systems of angle supports riveted to the inside of the shell may be used and may

suffice amply to support the brickwork when it is built into the furnace, but these narrow ledges cannot in my judgment furnish the sound and substantial support which a body of brickwork two feet or more in thickness receives by resting upon a solid foundation like the mantle.

Moreover the thinness of the inwall seems to have an influence upon the work of the furnace altogether apart from its effect on the durability of the lining. Thin-lined furnaces have been found by their managers to be sensitive to the quantity and temperature of the cooling water circulated over the shell and in general to have troubles to which furnaces with linings of normal thickness are not liable, and some of the best furnace managers in the country who have tried them have told me that they had had enough and were perfectly satisfied in the future to return to the standard lining.

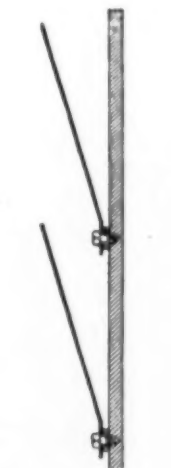


FIG. 12—
SPIRAL TROUGH
ON WARWICK
THIN-LINED
FURNACE

Some of the furnaces of the constructions here described are still in operation and some of them are said to be successful, but as a result of several years of experimentation no standard construction appears to have been developed which all advocates of this type of furnace are willing to adopt, and while the advantages of securing permanence of the lines of the furnace as constructed, of requiring for lining purposes only a small amount of brick, and therefore small expense and loss of time in relining, are undoubted, the essentially unsubstantial character of a wall only about a foot thick and 60 ft. or 70 ft. in height leaning inward at the top may well be questioned. Moreover, the thick lining has the advantage that within reasonable limits it gives the furnace some opportunity to cut itself out to the shape at which it will do the best work, so that minor errors of judgment in the original lines of the furnace as built may be corrected by the work of the furnace itself within the thickness of the standard lining, which is not possible with the thin one.

The lines of furnaces are not fixed like the laws of the Medes and Persians but shift with almost every campaign to meet changes of conditions or the changed views of their operators. With a lining of standard thickness and a standard shell large enough to contain it, such changes can within reason be made without affecting the shell, but with the thin-lined furnace this is not true. The shell must virtually have a shape exactly corresponding to that desired for the inside of the furnace and if it becomes necessary to change the latter this involves expensive and time-consuming changes in the shell.

For all these reasons, and for the far better one

mentioned above, that in practice the thin-lined construction does not seem to have justified itself, it seems doubtful whether the development in this direction will be extensive or at least very rapid.

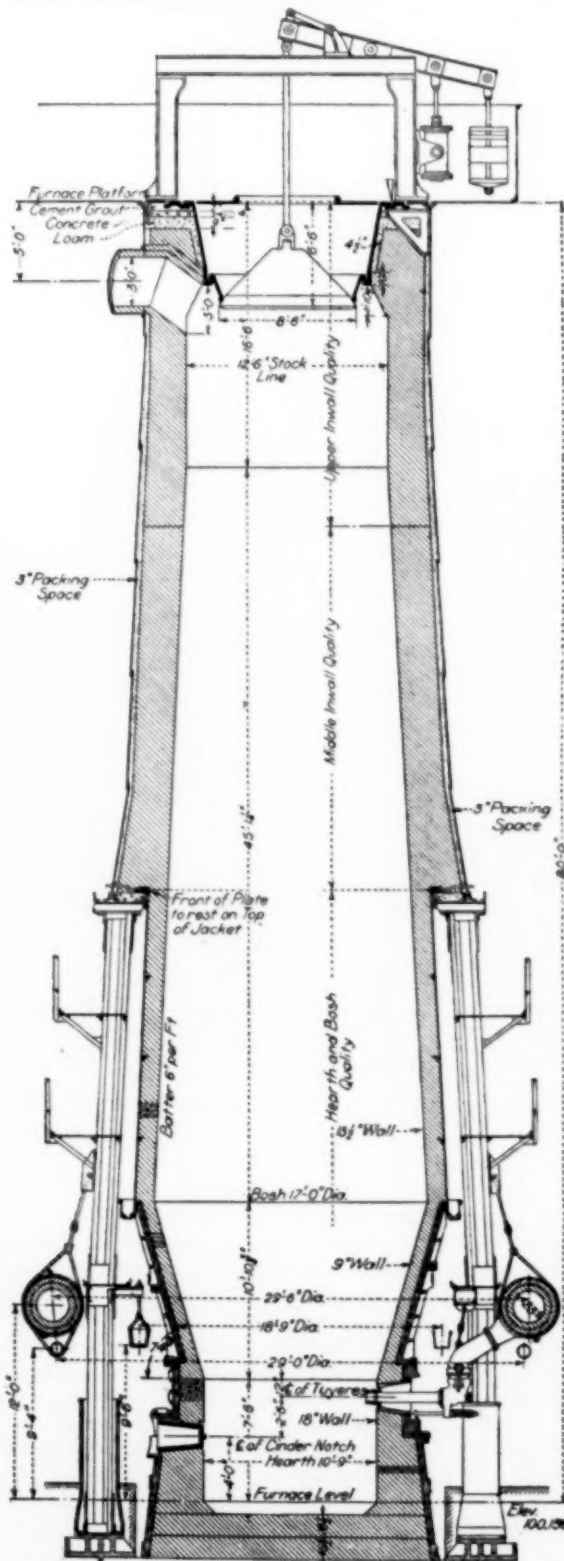


FIG. 13—SECTION OF NORTHERN IRON COMPANY'S PORT HENRY FURNACE

The Ashland Construction

The construction shown by Figs. 1, 2 and 4 in which the mantle is supported directly upon the tops of the columns, numbering in modern practice from ten to

twelve, has virtually been the standard ever since the adoption of the steel-shell construction, but is open to various objections from a practical and operating point of view.

These objections are that the columns come very close to the walls of the bosh and particularly to the outside of the hearth jacket, narrowing up the cooling ditch around the base of the latter and making the removal of any iron which may run into the ditch in a breakout extremely difficult, because this iron locks itself between the hearth jacket and the columns and in some cases can not be removed without blasting which would endanger the columns themselves. Moreover, when this construction is used the bustle pipe carrying the hot blast to the tuyeres has to pass outside the columns and the penstocks, carrying the hot blast from the bustle pipe to the individual tuyeres have to pass between the columns. These penstocks have to be swung out of the way when ever the tuyeres have to be changed, and the columns are greatly in the way at such times. There is nearly always a column close to the tuyere on one side or the other and access to the tuyeres is enormously hindered thereby. These columns interfere with the lead of the individual water pipes from the circular water main beneath the bustle pipe to the tuyeres and cooling plates and make it difficult to determine which cock controls which pipe, a matter of vital importance in case of a partial obstruction to the flow in one of these pipes, which then requires instant attention as the price of preventing the destruction of the cooling member which it supplies and the stopping of the furnace to replace the latter.

The writer having realized for years past the seriousness of these difficulties from an operating point of view when the charcoal furnace at Ashland, Wis., required reconstruction a few years ago, designed it along somewhat different lines, which are shown in Figs. 6 and 14. The central idea of the construction adopted is to remove the columns from their ordinary location immediately under the mantle and set them back far enough out of the way to give complete access to the base of the furnace and to prevent injury to the columns themselves in case of breakouts or other accidents around the furnace.

This was accomplished by building a square frame work of extremely heavy plate girders, the inside distance apart of which was slightly greater than the outside diameter of the shell of the furnace. These girders were supported on the top of the four columns to which they were riveted and strongly braced to prevent the possibility of failure by twisting. The shell was supported on these girders by eight brackets, two on each girder, these brackets being heavily riveted to the shell.

In order to prevent the sagging of the mantle plate eight cantilever brackets projected in under the mantle, one vertically under each of the main brackets. These

were supported by a downward extension of the main angles which fastened the main brackets to the shell and the outer ends were held down by struts extending down from the ends of the main brackets.

The construction was a success in all respects and gave a degree of accessibility and convenience in operation around the bottom of the furnace beyond comparison greater than that of any other type. This accessibility is indicated by Fig. 14, which shows two men abreast, standing shoulder to shoulder inside the column, and outside the outermost point of the penstock. Of course, the tuyeres do not come in line with the columns in this construction any more than in the ordinary one.

For a furnace of this size no change in the design of the construction need be made. For a coke furnace of greater size and with greater liability to serious difficulties of operation, it would probably be desirable to

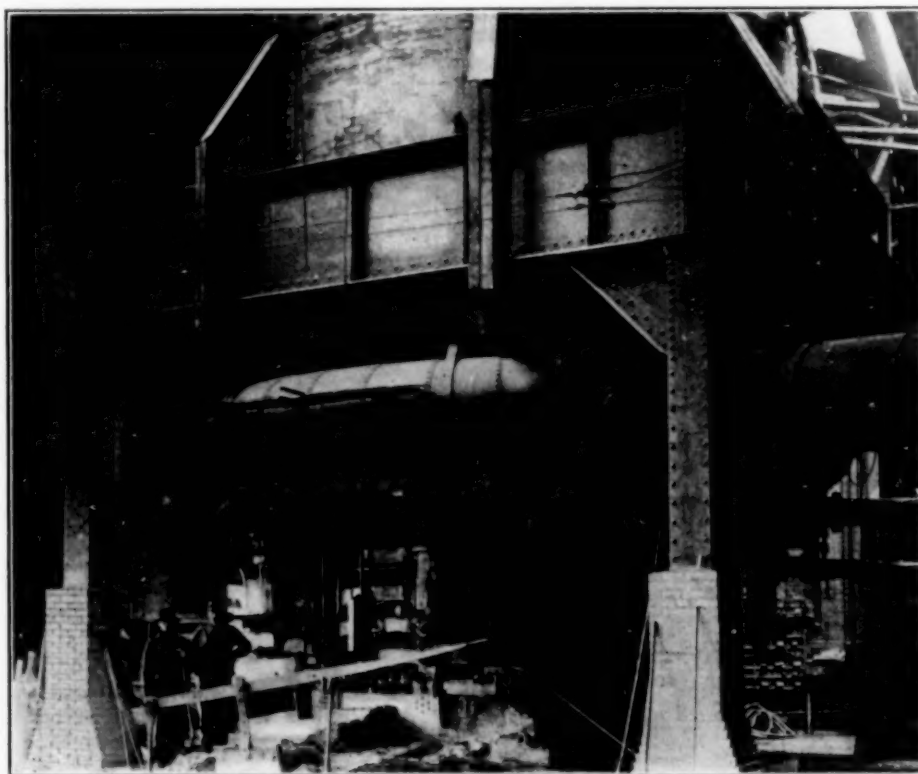


FIG. 14—ASHLAND CONSTRUCTION

use five columns, so that if one of them were to fail three of the girders, encircling more than one-half the furnace, would still stand and support it. This, of course, would only happen in case of a terrific disaster because the columns are set back so far from the furnace that no ordinary one could affect them, and on account of the abundant room around them they can be thoroughly protected by brick work.

In case this construction were used for a coke furnace a box girder would be used instead of the single plate girder and this box girder would be made watertight and serve for the circle pipe to supply cooling water around the furnace. This would prevent any possibility of the girders being softened or affected by heat no matter what happened.

This construction permits the bustle pipe to be put inside of the columns and thus to have the penstocks as close to the tuyeres as may be desirable without having any interference between the penstock and the columns. This also shortens up the bustle pipe materially, which reduces both its weight and its cost. One

of the important features of this construction when used to replace the standard type is that the foundations, lying entirely outside those of the old construction may be put in while the latter is being torn out, and the structural work may be erected before the old furnace is down, as was done in the case illustrated, with the saving of many weeks in time, worth many thousand dollars more than the slight additional cost of construction.

Firebrick

There is scarcely any feature of the construction and operation of the blast furnace of more importance than this and probably none concerning which less is known by the great majority of those to whom it is so important, the furnace managers.

The problem of firebrick has never been solved in terms of chemistry, or if so the solution is not known to furnacemen. This is probably because several other factors, such as the fineness of the grinding of the clay, and the method by which it is molded, the temperature and time during which the bricks have been burned, and other factors not reflected in their analysis have a vital influence on their character.

For this reason firebrick are to a great extent bought and sold on the basis of workmanship, freedom of the bricks from warpage, and general reputation. It is true that iron and alkalis in more than moderate amounts are barred because both unite with the silica at low temperatures and cause the fluxing of the brick under conditions far less severe than can be met with impunity by bricks free from these impurities. But further than this chemistry does not play a heavy rôle in the purchase or sale of firebrick.

It was customary a generation or more ago to use very large bricks, the size of 6 in. square on the small end, by 9 in. or 14 in. long was the standard in certain of the older iron-producing regions. But this was followed by a period in which a thickness of about $2\frac{1}{2}$ in. became standard, virtually the same as that of red brick. These were made in lengths of 9 in. and $13\frac{1}{2}$ in. so that two of one would equal three of the other. The straight, commonly known as the "square" brick, is $13\frac{1}{2}$ in. long, 6 in. wide, or in the case of the tapered or circle brick, known as "keys," 6 in. wide at the larger end. Similarly the straight brick, 9 in. long is $4\frac{1}{2}$ in. wide, and the "keys" $4\frac{1}{2}$ in. wide at the large end.

These are relatively small bricks and require much labor for laying and involve a great number of joints, but it was claimed by some furnace men for many years that thicker bricks could not be burned satisfactorily, though I have seen years ago thousands of brick 6 in. thick which gave complete satisfaction in the furnaces in which they were used, which were small coke furnaces.

In very recent years firebrick manufacturers began to make furnace bricks 3 in. thick, and these are very much freer from warpage in the burning process than the thinner bricks as well as requiring less joints and less labor to lay them. They are, therefore, very much to be preferred from practically all points of view except that of cost.

In this respect the brickmakers have established certain customs which have not been permitted to any other trade. The standard price of bricks is based on the 9-in. by $4\frac{1}{2}$ -in. by $2\frac{1}{4}$ -in. brick. Smaller bricks than the 9-in. are charged for at the same rate per brick as if they were the 9-in. size. This is explained by the brick maker on the ground that the clay in the brick amounts to very little but that the workmanship on it is just as much in the small brick as it is on the large one. The $13\frac{1}{2}$ -in. bricks, on the other hand, take a

price of nearly 20 per cent higher than the 9-in. based on the same volume.

When he is asked why this is so the brick maker replies that they are more difficult to handle and to burn.

Similarly when it comes to the 3-in. lining instead of the standard thickness, the brick maker insists that he must have an increase of about 20 per cent in the price per unit of volume of these, although the warpage in this case is very much less, the labor decidedly less, and the burning but little more, apparently on the ground that he needs it, and he usually succeeds in getting it.

It is, however, no more than fair to the brick manufacturer to say that quality does not always receive the consideration from the furnace man that it should, and the manufacturer of a superior grade of brick is often forced to compete against a poor grade of brick on a piece basis, which is neither fair nor sensible, but is to a large extent the outcome of the difficulty in determining what real quality is, already described.

Firebrick are made in three qualities known respectively as "hearth and bosh," "inwall," and "top," intended for use in those regions of the furnace for which they are named, except that formerly the "hearth and bosh" quality was usually continued a number of feet above the top of the bosh, but in the best recent practice this quality stops at the top of the bosh.

These differences in quality are adapted to meet the difference in conditions in the different portions of the furnace. In the hearth and bosh there is little or no abrasion but intense temperature, and intense scouring action of slag and iron. To meet these conditions the bricks are made with a relatively large proportion of flint clay and a correspondingly smaller proportion of plastic clay. These bricks are not exposed to much abrasion and are therefore not particularly designed to resist, but through their composition and burning are made to stand extensively high temperatures successfully.

In the middle zone of the furnace in which the lining is known as the "inwall" the conditions of temperature and chemical action are considerably less severe, while those of abrasion due to the presence of some of the solid portions of the charge and the sand-blast action of its fine portions at the velocity imparted by the blast are considerable, and the cone in this portion of the furnace being convergent upward the sand blasting action is a factor to be reckoned with, whereas in the bosh with its divergent upward flare this is not the case. Inwall bricks, therefore, are made with more plastic and less flint clay than hearth and bosh brick to meet these conditions. At the same time they still have high heat resisting qualities.

At the top the conditions of abrasion are extreme, as has already been described, while the temperature in good practice in this zone of the furnace is generally below 1000 deg., so that the necessity of refractory character in the lining of this zone can be to a certain extent neglected, or at least need only receive an insignificant amount of attention as compared with what it demands in the lower zones. For this purpose the bricks are made with a large proportion of plastic clay pressed very hard, and burned to a great density, some of them resembling what is known as vitrified paving brick, almost as much as they do firebrick, and for the same purpose, resistance to abrasion.

Where no form of stock line protection is used such as those described above or an equivalent, the quality of the top brick is fully as important as that of the hearth and the bosh brick since top bricks which do not resist abrasion, permit the stock line to be gouged out by the drop of the stock from the bell, with the result that the correct distribution of the stock is made

impossible and becomes largely haphazard, which is necessarily followed by extremely bad operation in the lower portion of the furnace, with the result of poor quality of iron, high fuel consumption, or of both. The necessity of repairing the stock line has for this reason been responsible for an appreciable percentage of all the cases in which blowing out for repairs have been necessary, though the fact may not have been apparent in all cases, because when a furnace which has been working well gradually begins working worse and worse, it is commonly and in general rightly assumed that the lining is too far gone to permit it to do further good work, and without any analysis as to the location of the wear which is responsible for the bad work, it is generally attributed to the inwall, although the top may be the part to blame.

It is unfortunate that more exact and accurate knowledge is not at hand on the subject of firebrick and their suitability for lining purposes, but the problem is so complicated that a complete analysis would be possible only after years of experience by an organization which contained both an experienced firebrick maker and experienced furnace men, a condition which has seldom if ever existed.

The firebrick manufacturer has, therefore, to adapt his product to the conditions described by the furnace man, and the furnace man has to use the brick furnished to him by the brick maker in the prayerful hope that they will give the results desired, but in the vast majority of cases without any better foundation for this hope than his confidence in a given brand of brick, and perhaps his belief that a certain brick maker understands the conditions well enough to provide him with a suitable lining.

It is greatly to be desired that the furnace man and brick manufacturer should meet directly rather than through their respective selling and purchasing agents. By discussing frankly and without prejudice their respective objects and difficulties progress would be made in real knowledge of this subject with resultant benefit to both.

I have already described the enormous change in the furnace conditions which began in the middle nineties. The greatly increased outputs and the vast increase in percentage of fine ore used in the burden, in combination with the introduction of mechanical charging brought about conditions under which the lining problem seemed to be almost insoluble. I know on the best authority that one of the largest brick makers was desperate at one time over the situation and would have been glad had he felt that he could afford to do so to surrender the furnace-lining portion of his business.

It seems extremely likely from the information which has been developed within recent years in connection with methods of charging, that these have been more responsible for unsatisfactory life of linings than was the poor quality of the linings themselves in many cases in which the firebrick manufacturer had to take the blame.

It has long been known that bad filling would rapidly destroy a lining by channeling, but certain types of mechanical charging which are theoretically correct produce conditions in practice under which it would be impossible for any lining to survive for any considerable length of time.

This matter really belongs under the subject of stock distribution and will be treated at greater length in an article on that subject.

THE LAYING OF THE FIREBRICK

The mortar used for this purpose consists of ground fireclay which is mixed to grout of a consistency to suit

the ideas of the furnace man or the boss bricklayer, and usually kept hot during the mixing and to a considerable extent even during laying, by a steam coil in the bottom of the clay box.

This grout or mortar is generally mixed quite thin, poured onto the brickwork with a large dipper and flushed into the joints of the brickwork, both from below by sliding the brick being laid up against the one just laid, so as to fill the joint as nearly as possible with fireclay, and then from above by filling in the space and working the fireclay down between the two bricks with a trowel if any opening sufficient for that purpose exists, which in general it should not do.

Many of the difficulties with linings have been attributed by some furnace men to bad bricklaying and particularly to wide joints. Some go so far as to insist that all unevenness must be removed from the bricks by chipping each individual brick where any appreciable amount of warpage exists, producing what is known as a case-knife job. This means that every joint shall be laid so close that it is impossible to get the point of a table knife into it.

Thick joints are undoubtedly very bad, but I have never seen any results produced by such extreme precautions which seem to me to justify the very greatly increased expense in laying the lining by this method. Careless work should on no account be tolerated, and if joints are found which are not thoroughly flushed the brickwork should be torn down and rebuilt properly. But as long as the firebricks themselves are made of clay suitably burnt, it has never been evident to me why the clay in the joints should not be burnt practically into the same structure as the brick itself, and in fact the brickwork which is torn out during the relining periods from the lower portion of the furnace where the heats have been high come out in blocks, the line of whose breakage frequently pays little or no attention to the original joints in the brickwork, proving to my mind that when properly laid, the fireclay joint becomes, by the action of the furnace itself, almost as good as the brickwork.

This statement is, of course, predicated on the assumption that fireclay of the same quality as the brick will be used. If inferior clay be substituted it may act as flux for the brick themselves and absolutely destroy the lining.

The practice of making the fireclay thin may easily, in my judgment, be overdone, for the reason that if the clay is very thin, when the bricks are settled to place with a mallet, if there is the least particle of warpage, the brick springs enough to throw the thin clay out of the joint and when it springs back to shape again this joint is practically open. When the fireclay is made so thick, on the other hand, that it will just about pour from the dipper, it is too plastic to permit this action and while the bricks may be driven up to a solid bearing with one another the slight joint which remains after they are in contact is fairly filled with fireclay.

THE FILLING OF THE PACKING SPACE

In former times when furnace shells were built not much more than a quarter as strong as they are to-day they were frequently burst by the expansion of the brickwork within them and it was therefore universal practice at one time to provide a space of 3 in. to 6 in. between the outside of the brickwork and the inside of the shell which was filled with some yielding material to allow for the expansion of the brickwork, and yet continually resist it. In modern practice shells are so strong that they can resist the expansion of the brickwork and the latter is therefore laid up close to the shell and the small space necessary as a construction

allowance is quite frequently grouted with concrete which, of course, is entirely unyielding. This means that the shell must be strong enough to resist the expansion of the brick or else it must stretch enough to accommodate the expansion of the brick within its elastic limit. Probably both these actions take place to some degree, but in any event this plan is not safe except with a shell of the strongest and best construction. If the brickwork were laid up tight inside a shell of older type the latter would probably be burst within a short time, as unfortunately has happened only too often even with the greatest precautions.

To prevent this it is customary with such shells to leave a space between shell and lining and fill it with some material which is yielding and yet to some extent resistant. The necessity of resistance arises because if an open space were left between the brickwork and the shell the internal pressure would push the brick back, break the bond, and ruin the keying, so that some bricks would be likely to fall out which would start a hole in the lining, and once started this would rapidly enlarge itself so as to destroy the lining. The bricks therefore must be held up to their work by some sort of filling between them and the shell.

Some furnace men prefer for this purpose clay, but clay is only slightly compressible, and I prefer pumice cinder, made by running a light slag onto wet ground, or into a tank of water. The steam formed by the heat of the slag puffs the latter up into a huge sponge whose total volume is many times the volume of the slag from which it is made. As it is crushed under pressure its volume becomes smaller and smaller by the progressive destruction of the cells, first of the larger and then of the smaller sizes, but the material maintains its resistance to compression almost constant throughout a very great diminution of volume.

These are the conditions which we have to meet, as brickwork, unless rigidly bound, does not expand elastically like steel, but when expanded retains its new size, never contracting, or never to anything like the degree to which it had previously expanded, so that as its temperature varies it undergoes a slow but constant growth. Thus a furnace shell may be amply large at the beginning of a campaign and for several months or a year, but gradually as the lining increases in size, when all the compressibility of the backing is gone, the direct expansionary force of the bricks is transmitted to the shell, and while the steel is stronger than the brickwork inch for inch, the latter is so much greater in area in any given section of the furnace that it generally bursts a thin shell when the "give" of the backing is gone.

In order to make the backing gas-tight the pumice cinder is sometimes mixed with clay, but this has the effect of reducing its compressibility especially as the clay is apt to burn hard after the furnace has been in operation for a while, and this therefore limits the amount of compression which a given portion of the backing can undergo without exerting undue stress upon the shell.

DRYING THE LINING

Even with the best possible practice several tons of fireclay are used in laying the lining of a furnace, and this being mixed with something like an equal weight of water there is a great quantity of water in the lining when the brick laying is finished. This is capable of doing great damage and should therefore be removed by thoroughly drying the lining before the furnace is put into operation.

Various methods are in use for this but usually a temporary brick oven is built immediately in front of the tapping hole, and communicating with the furnace

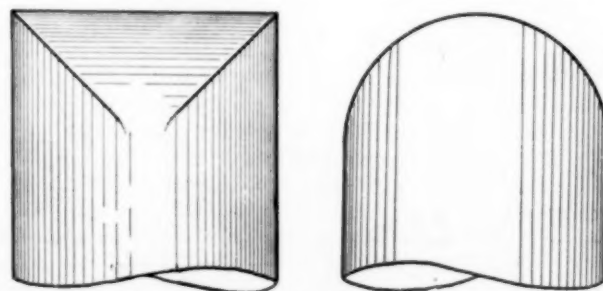
through it. The bell is closed and draft produced either by opening the bleeder or allowing the main draft stack to draw on the furnace through a stove. A fire is built in the temporary oven and the gas from this passes into the furnace and warms it up, drawing out the moisture from the face of the brickwork first, and gradually as the heat works through drying out the whole mass quite completely.

The hot gas also acts as a carrier for the steam formed from the moisture and takes it out of the furnace. This drying-out process lasts anywhere from one to three weeks according to my judgment of the furnace manager and the state of the iron market.

There is no doubt that the operation is much better carried on slowly than rapidly, but a very moderate rate of firing will bring the warmth through a wall of 2 ft. or 3 ft. thickness in the course of three or four days, and a week of this treatment is in general sufficient to dry out the clay about as thoroughly as it can be done by this method. Of course, if the furnace is ready for operating before the rest of the plant it is well worth while to continue the fire for drying out for two or three weeks or even longer if convenient. But in my judgment three-fourths of all the benefits to be obtained by drying will be obtained in the first week of moderate but steady firing, day and night.

The removal of this moisture not only prevents the possibility that the rapid heating which would come from the actual operation of the furnace on a green lining would convert the moisture of the fireclay into steam so fast as to drive the clay out of the joints, and perhaps blow objectionable holes through the packing at the back, but it also exercises a very objectionable effect on the operation of the furnace while blowing in when it is already very cold by reason of the fact that its initial charge is not preheated in its descent, but necessarily cold clear to the bottom.

I have never had any experience with blowing in on a green lining, but I should expect that much more fuel would be required for the first two or three days of such blow in than would be required on a furnace with a thoroughly dried and warmed lining, and that



CYLINDRICAL STOVE TOP

the difficulty and delay in getting the furnace into normal operation could be much greater.

Erratum.—In our last issue, page 408, Fig. 35 of a cylindrical stove top was incorrect due to a mistake of the draftsman. It is now reproduced in correct form. In the early days many stoves were fitted with a cylindrical top as shown in this illustration and this shape conforms with the mechanical requirements that all sections of the stove shall be circular in order to resist deformation, but this style of top is difficult and expensive to fit and seems particularly liable to give out along the square corner where the horizontal cylindrical surface of the top meets the vertical surface of the stove body, and stoves are now almost universally fitted with either conical or spherical tops.

Notes and Observations on Ore-Treatment at Cobalt, Ontario, Canada—II.

BY H. C. PARMELEE

Concentration

In a preceding article¹ the writer pointed out the growing importance of concentration at Cobalt, and showed that by far the greater tonnage of ore mined is treated first by ore-dressing processes. The material thus treated is low-grade wall rock, containing from 15 to 30 oz. silver per ton. It is distinguished from the high-grade ore which occurs in narrow veins and contains from 2000 to 4000 oz., and even as high as 6000 oz., silver per ton. The value of the low-grade ore depends on the extent to which the vein walls are mineralized. As delivered to the mill it is enriched by the admixture of fine ore broken from the high-grade veins. Mill feed in the district will range from 20 to 35 oz. silver per ton.

By preliminary processes of hand-sorting, either underground or on picking-belts, and coarse crushing and jigging, the high-grade vein material is readily recovered. The wall rock, on the other hand, in which

extended flow sheet published by him.² This shows at a glance several features which impress the visitor at Cobalt mills: (1) the general use of stamps, and (2) the lack of stage crushing and concentrating.

Crushing and Grinding

The use of stamps or rolls on Cobalt ore is a point of controversy among millmen in the district, although generally the seal of approval is placed on stamps if we may judge by their more extended use. Fig. 1 shows that stamps are used for the crushing of slightly over 80 per cent of the total tonnage concentrated. An objection to stamps is the quantity of slime and fine sand produced, amounting to 30 or 40 per cent of minus 200-mesh pulp when crushing through 16 to 30-mesh screen. This objection is emphasized in those cases where no regrinding is practiced, and where fine screens are used on the batteries to insure original grinding to the requisite degree of fineness. It is claimed that rolls will make not more than half the amount of fine produced by stamps. The Chilean mill also slimes less than stamps: thus, a feed of $-6 + 12$ mesh material ground in a Chilean mill through 12-mesh screen gave only 6 per cent minus 200-mesh.

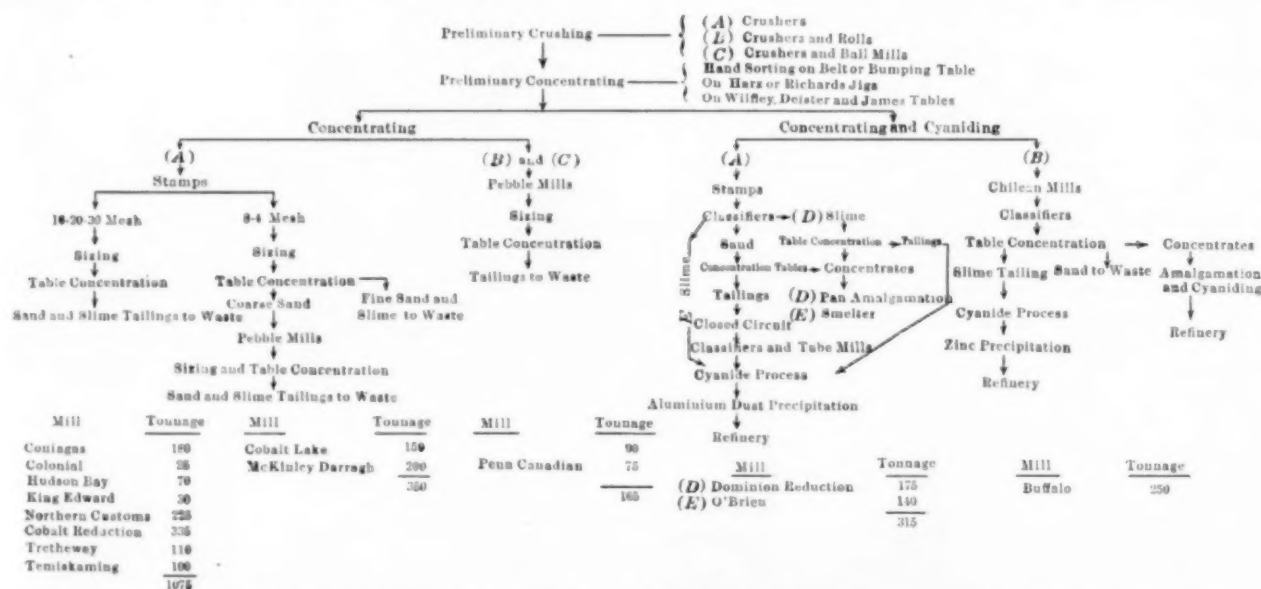


FIG. 1—COMPARATIVE FLOW SHEET OF COBALT CONCENTRATORS (FROM DATA COMPILED BY FRAZER D. REID)

silver and its minerals are disseminated, requires fine crushing before concentration can commence.

Excepting the friability of the various silver minerals, the low-grade ore presents no serious difficulties in concentration. The effect of an increased proportion of these minerals in the ore is to raise the grade of slime tailing, and consequently increase the loss unless this product is cyanided. Thus, at one of the mills where cyanidation of slime tailing is practiced, when the concentrator was treating a 40-oz. ore the slime tailing carried as much as 60 oz. per ton, but when the grade of mill feed dropped to 20 oz. the slime was reduced to about 8 oz.

The ratio of concentration varies at the different mills, but is high. An average probably would give between 45:1 and 50:1, with an extreme range of 25:1 and 100:1.

A comparative flow sheet of Cobalt concentrators, including those at which cyanidation also forms a part of the treatment, is given in Fig. 1. This is compiled from data contributed by Mr. Fraser D. Reid, superintendent of the Coniagas mill, and is abridged from a more

The advocates of stamps claim that rolls are inefficient machines in crushing the tough conglomerate; and even some of those who favor rolls, when condition of product alone is considered, admit that rolls do not stand up to the work and that crushing cost is increased by their use. At the same time it is probable that large heavy rolls have not been given a trial. Another argument in favor of stamps is that the ore can be ground in them to the requisite degree of fineness in one operation, thereby eliminating regrinding machinery. This leads, of course, directly to a related point on which also there is a difference of opinion.

Stage Grinding and Concentrating

To anyone who is familiar with general concentration practice on base-metal ores, such as occur in the western United States, the absence of stage reduction and concentration in most Cobalt mills will be a matter of surprise. The general tendency is to crush in one operation to a degree of fineness which experience shows will liberate most of the valuable metal and minerals. There

¹This Journal, July, 1914.

²Milling Practice in Cobalt, by Fraser D. Reid, *Bulletin, Can. Min. Inst.*, May, 1914.

are exceptions, notably the McKinley-Darragh and Cobalt Lake mills, wherein the crushing is done in stages by successive machines, with intermediate classification and concentration.

It is true, as already pointed out in our last issue, that all of the silver in these ores will not be released from gangue save by very fine grinding, and the millman must determine the economic limit of crushing. This seems to have been found between 16 and 30 mesh at different mills. Those who advocate single-stage crushing maintain that by grinding to, say 20 mesh and concentrating, a 16 to 20-oz. ore will yield a tailing carrying only 3 to 4 oz. silver per ton. To make a further recovery from this tailing would require very fine grinding which would not be profitable. In support of this view, Mr. Reid gives the following screen analysis of a tailing obtained from such treatment, showing the uniform silver content of the different grades down to minus 200 mesh, where there is a sudden increase.³

Screen Analysis of Sand Tailing from 16-Mesh Battery Discharge

Mesh	Per Cent Weight	Oz. Silver per Ton	Per Cent Value
+ 30	22.1	3.2	22.6
+ 60	31.8	3.0	30.3
+ 100	28.6	3.0	27.4
+ 200	13.1	2.5	10.5
- 200	4.4	6.5	9.2

Opposed to single-stage grinding are those who feel that the concentrating practice can be improved by crushing in several stages, with suitable classification and concentration at each step. While the advocates of this procedure are in the minority, they feel that progress in concentration in the district should be along this line and that future economic conditions will demand a change in this direction. Perhaps a suitable line of crushing and grinding machinery for this work would be gyratory and jaw breakers, large heavy rolls, disc grinders and pebble mills.

A new tube mill liner, known as the Globe, is being used at the McKinley-Darragh mill and is regarded favorably. Like the El Oro it consists of metal ribs



FIG. 2—CONIAGAS MILL, COBALT

with spaces in which the pebbles become wedged, but the ribs extend around the inner face of the tube instead of lengthwise. This liner is being used also at the Hollinger mill in the Porcupine district.

With regard to classification, it may be said that it is not practiced to the degree considered advisable and essential elsewhere. Perhaps the high specific gravity of the concentrate makes this necessary, but it is certain that more extensive classification is now being adopted in some plants, and the operators of those mills feel that the change is showing an improvement in results.

³Ibid.

Harz and Richards jigs are used for coarse concentration, but the former predominate. They are relied upon to recover silver nuggets and high-grade ore, which ordinarily are removed by skimming the jig beds. Wilfley, Deister and James tables are found throughout the district, the two latter being in favor and exceeding the first in number.

The Coniagas Mill

One of the typical concentrators of the district is the Coniagas, a view of which is given in Fig. 2. The ore is crushed in stamps, classified and concentrated, the tailings going to waste. Average mill feed contains 34 oz. silver per ton; sand tailing, 3 to 4 oz., and slime tailing, 6 to 7 oz. Preliminary treatment ahead of stamps results in cutting the grade of original feed in

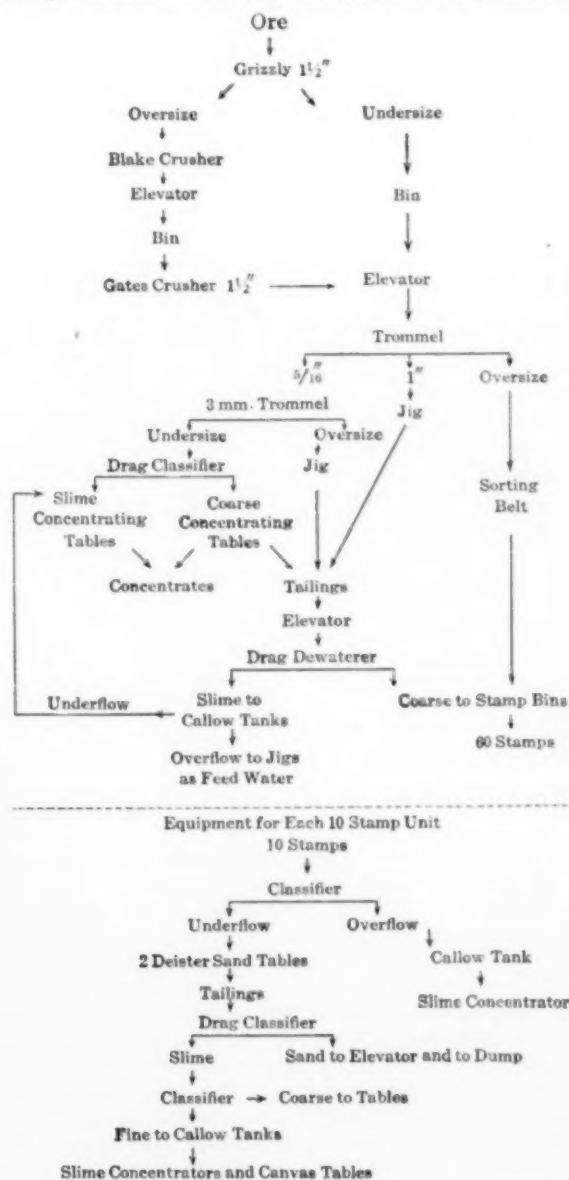


FIG. 3—FLOW SHEET, CONIAGAS MILL, COBALT

half, so that the stamp discharge contains about 16 oz. silver per ton. Concentrates are shipped to the company's smelter at Thorold, Ont. A flow sheet of the mill is outlined in Fig. 3.

Manganese and chrome steels are used in crushing and grinding machines and wherever apparatus is subjected to excessive abrasion by the ore. Jaws and cheek-plates on Blake crushers, and concaves and mantles on gyratories are manganese steel. The same metal

is used for buckets on dry and wet elevators. Buckets on dry elevators have been in service for three years, handling 180 tons of ore per day, and probably are good for the remaining life of the mine. On wet elevators these buckets have lasted for 15 months and probably will show a life of two years. Trommels also are of manganese steel, as are the mortar liners. The latter are 1 inch thick and sectionalized so that the part most subject to wear can be reversed and then replaced without discarding the entire plate. This practice has resulted in doubling the life of the liners.

Tailings Disposal

The Coniagas mill is so situated that sand tailings have to be stacked. As shown in the flow sheet the sand and slime are separated by a drag classifier, the slime being concentrated further and the sand sent to the tailing elevator, which is shown at the left in Fig. 2. Sand is discharged from the elevator into a launder with the minimum amount of water required for transportation and flows to the point of discharge where it stacks as shown in Fig. 4. The cost of this method of



FIG. 4—SAND TAILINGS STACKED AT CONIAGAS MILL

stacking is 0.7 cent per ton. A sample of the tailing is cut out at the elevator discharge and returned to a pilot table in the mill, where the attendant is able to observe the presence of any metal or mineral that may be going out of the mill due to faulty operation.

Another system of stacking sand tailing, by aerial tramway, is shown in Fig. 5. This is the method



FIG. 5—STACKING SAND TAILINGS AT BUFFALO MILL, COBALT

adopted at the Buffalo mill. The sand is dewatered in the mill by a drag conveyor which discharges into a bin from which the tram buckets are loaded. There are two buckets attached to the running cable, one being at the loading station while the other is dumping. The buckets dump automatically and the sand is sluiced down the launder by water conducted to the dump through a pipe shown in the view.

The Trethewey Mill

In Fig. 6 the treatment given at the Trethewey mill is outlined in considerable detail. This plant comes within the same classification as the Coniagas, stamps being used for crushing, followed by simple classification and concentration on sand and slime tables. There is no regrinding. At one time a Huntington mill was used as a regrinder, but it was abandoned as being wholly unsuited to the work of grinding such tough ore. James and Deister tables are used, and a Deister-Reid slime table recently has been added. This latter is a modification of the Deister slime table by Mr. Reid, superintendent of the Coniagas mill, and is finding considerable favor in the district.

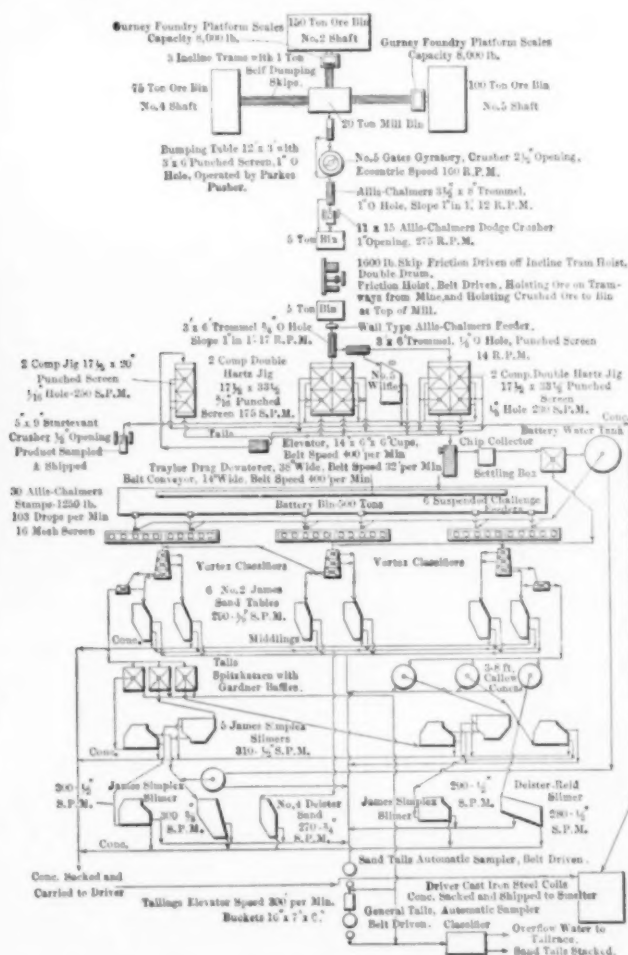


FIG. 6—FLOW SHEET, TRETHEWEY SILVER COBALT MINE, LIMITED

A successful device for removing chips of wood from the ore before it passes to the stamps is in use at this mill. It is placed at the drag dewaterer which receives the jig tailings. It consists of a series of brushes mounted on a belt traveling across the trough of the dewaterer at right angles to the travel of the drag. The brushes dip beneath the surface of the water, engage the floating chips and remove them at one side. The device has been of great assistance in avoiding the blinding of battery screens and preventing the accumulation of fine wood pulp in the concentration system.

Cost of Concentration

Mr. Reid gives the cost of concentration at the Coniagas mill as \$1.32 per ton of ore treated. This excludes the cost of coarse crushing and preliminary concentration, and refers only to the work done at and

*Ibid.

beyond the stamps. It includes, however, allowances for depreciation, interest on investment, insurance, etc. At another mill the cost of all work, including coarse crushing and concentration, is given as \$2.67 per ton. Details of various concentrating mill costs given by Mr. Reid are shown below.

Coniagas Mill.—Capacity, 180 tons. Sixty stamps and concentrating equipment cost \$75,000. Battery feed is 2-in. size and finer, and is crushed through 16-mesh screen. Assay of battery discharge is 16 oz. silver per ton. Recovery is 11.8 oz.; value, @ 55 cents, \$6.49.

Stamping:	Cost per Ton
Power	\$0.157
Operating072
Shoes022
Dies011
Screens004
General016
	\$0.282
Concentrating:	
Labor109
Power037
Supplies025
General008
Fuel, etc.032
	0.211
Marketing, 11.8 oz. silver at 4.6c.....	0.523
Depreciation, 10 years.....	0.138
Interest on investment, 6 per cent.....	0.046
Insurance, 2.2 per cent.....	0.030
Assay loss, allowance 1.34 per cent.....	0.087
Total.....	\$1.317

Beaver Mill.—Capacity, 80 tons. Pebble mills and concentrating equipment cost \$50,000. Assay of mill feed 19.5 oz. silver per ton. Recovery is 15.65 oz.; value, @ 55 cents, \$8.61.

	Cost per Ton
Marketing, 15.65 oz. silver at 5.9c.....	\$0.923
Working costs	1.290
Depreciation, 10 years.....	.198
Insurance, 2.2 per cent.....	.043
Interest on investment, 6 per cent.....	.090
Assay loss, allowance 1.34 per cent.....	.115
Total.....	\$2.659

In another mill of 200 tons capacity, the cost of which was \$125,000, the mill feed assays 30 oz. silver per ton. Recovery is 25.5 oz.; value, @ 55 cents, \$14.025. Costs include preliminary crushing and jigging.

	Cost per Ton
Operating	\$0.292
Power and light.....	.250
Repairs152
Management120
Steel072
Fuel046
Assaying and sampling.....	.043
Screens and general018
Oil009
	\$1.002
Depreciation, 10 years.....	.201
Interest on investment, 6 per cent.....	.060
Insurance, 2.2 per cent.....	.044
Assay loss, allowance 1.34 per cent.....	.188
Marketing, 25.5 oz. silver at 4.6 per cent.....	1.173
Total.....	\$2.668

Cyanidation of Cobalt Ores

It can be readily understood that the proposal to reduce the complex ores of Cobalt by the cyanide process was received with skepticism. Consequently the successful application of this method may be regarded as a triumph in hydrometallurgy, as pointed out by Mr. Clevenger in our recent symposium on the complex-ore problem.⁵ Containing, as they do, combinations of silver, arsenic, sulphur, antimony, cobalt and nickel, these ores do not appear attractive to the cyanide metallurgist, and it is not surprising that concentration and smelting have formed the predominating basis of treatment. Advocates of the latter system claim an advantage on low-grade ore, where the stamp discharge carries from 12 to 15 oz. silver per ton after about an equal amount has been removed by preliminary coarse crushing, sorting and jigging. Possibly the margin of profit in favor of either process is narrow in any event, but when the Nipissing company contemplated the erec-

⁵This journal, May, 1914, p. 300.

tion of a low-grade mill for the treatment of, say 30-oz. ore, and considered the merits of concentrating and cyaniding, the latter process was adopted. Preliminary estimates showed a profit of \$1.59 per ton in favor of cyaniding,⁶ and the results since obtained in practice have more than confirmed these estimates. Certainly the advantage gained by a quick conversion of ore into marketable bullion is a considerable benefit offered by cyaniding as compared with concentrating and smelting.

Excluding those mills at which slime tailing from concentration only is cyanided, there are others in which cyanidation, with and without amalgamation, is the reduction process employed. At the Dominion Reduction and Buffalo mills the ore is first concentrated, after which the slime tailings are cyanided and the concentrates treated by a process of cyanide-amalgamation. At the Nipissing high-grade mill⁷ cyanide-amalgamation is employed, while at the same company's low-grade mill cyanidation only is used. The process of cyanide-amalgamation in use at the Buffalo is the same as that first used at the Nipissing high-grade mill.

The Nipissing Mills

The Nipissing Mining Company operates two mills: one for high-grade ore, which has been in operation since February, 1911, and the other for low-grade ore, which went into commission in November, 1912. Since the construction of the latter mill all ore is received at a central washing and sorting plant, where the rich ore is selected and sent to the high-grade mill, the balance going to the low-grade plant. The washing plant and low-grade mill are situated on Nipissing hill, across Cobalt lake from the town of Cobalt. A view of this mill, as seen from the town, is shown in Fig. 7. The high-grade mill is shown in Fig. 8, and a flow-



FIG. 7—NIPISSING LOW-GRADE MILL, FROM TOWN OF COBALT

sheet is given in Fig. 9 showing the relation of the two plants.

Washing and Sorting Plant

Run-of-mine ore contains, on an average, 50 to 55 oz. silver per ton. It is crushed to 3-in. size before delivery to the bins of the washing plant, where it is drawn into cars and weighed on registering scales that give an accurate record of all ore received from the mines. In the succeeding operations the ore is washed, sized, hand-picked and jigged for the removal of the high-

⁶Mill and Metallurgical Practice of the Nipissing M. Co., by James Johnston, January, 1914, *Bulletins Can. Min. Inst.* and *Amer. Inst. of Min. Eng.* Abstracts in this journal, March, 1914, p. 189, and May, 1914, p. 348.

⁷Nipissing High-Grade Ore Mill, Cobalt, by R. B. Watson, *Eng. & Min. Jnl.*, Dec. 7, 1912.

grade portion, containing about 2500 oz. silver per ton, which is sent to the high-grade mill. The jigs handle two sizes of ore, one between 0.5 and 1.5 in., and the other between 3 mm. and 0.5 in. The discarded low-grade ore coarser than 1.5 in. is crushed to that size before going to the battery bins; the balance, including jig tailings, is sized through 3-mesh screen, the oversize going to battery bins and the undersize to the tube mills. In this way the batteries are relieved of handling about 10 per cent of the ore which is already crushed fine enough for tube mill feed.



FIG. 8—NIPISSING HIGH-GRADE MILL, COBALT

Cyanide-Amalgamation of High-Grade Ore

In 1913 the high-grade mill treated 1832 tons of ore containing 4,804,928 oz. silver, of which 1200 tons was Nipissing ore averaging 2501 oz. per ton. The tonnage treated at the high-grade mill is between 1½ and 2 per cent of the total ore mined and sent to the washing plant. The recovery is 99 per cent, of which 97 per cent is by amalgamation and 2 per cent by cyanidation.

The main step in the process, which was devised by Charles Butters and G. Howell Clevenger, consists in grinding the ore for 10 hours in a tube mill with mercury and strong cyanide solution. The ore is first crushed to 20-mesh size. A tube mill charge depends somewhat on the grade of the ore, but consists in round numbers of about 3 tons of ore, 4 tons of mercury, 1½ tons of 5 per cent cyanide solution and 6 tons of pebbles. The tube mill is of special construction and is adapted to receive compressed air to aid in cyanidation. Fine grinding is necessary to liberate all of the silver, and at the end of the operation practically all of the pulp will pass a 200-mesh screen.

The tube is charged and discharged through manholes. When the pulp is ready to be discharged the manhole covers are replaced by screens through which the pulp and amalgam flow into a settling tank. Amalgam and mercury are drawn from the settler into a clean-up pan and then into canvas filters, and the amalgam is finally retorted.

The ore residue is given a further treatment by agitation in cyanide solution, followed by decantation and vacuum filtration. The solution is precipitated by zinc shavings in the usual manner. An unusual sight is presented in the zinc boxes owing to the change of color of the solution as it flows from upper to lower compartments. The head solution is clear and colorless, but it gradually acquires a reddish tint and exhibits a marked coloration in the last box. This is due to the presence of nickel in solution and probably results from the reduction of nickel cyanide by zinc. When this colored solution is again oxidized by aeration it resumes a normal appearance.

Recovery of Cobalt Residue and Silver Bullion

The filter residue contains mainly cobalt and nickel minerals, some silver and mercury. The total consumption of mercury in the entire operation of producing bullion is about 20 lb. per ton of high-grade ore, of which about 12 lb. remains in the residue. The balance is lost in volatilization and handling. The residue carries also 20 to 40 oz. silver per ton, 8 to 10 per cent cobalt, 4 to 6 per cent nickel and 30 to 40 per cent arsenic. It is sold to manufacturers of cobalt products, who pay for the cobalt and 85 per cent of the silver. In 1913 the company sold 1659 tons of residue, netting \$62,484.

Retorted amalgam and zinc precipitate are melted in a reverberatory furnace and oxidized with air to slag off impurities. The resulting bullion is 999 fine. A small blast furnace is used periodically to smelt rever-

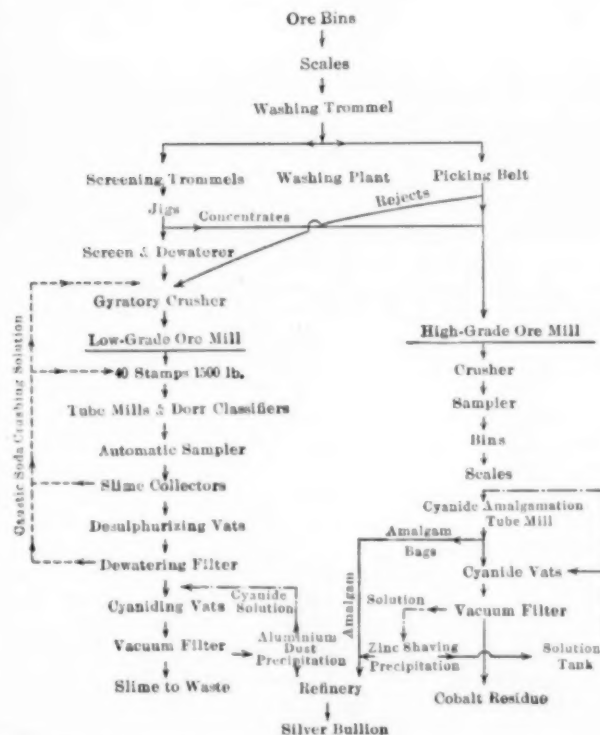


FIG. 9—FLOW SHEET, NIPISSING MILLS, COBALT

beratory skimmings and flue dust, and the resulting bullion is further refined in the reverberatory furnace. A baghouse has recently been constructed to recover dust and fume from the blast, reverberatory and retort furnaces, thus adding another refinement to a process which yields a commercially complete recovery of silver from a complex high-grade ore.

Interesting and successful as the Nipissing high-grade mill has been, it is possible that the process may be supplanted by direct cyanidation. Research, which has been such a prominent feature of Nipissing metallurgy in the past, is constantly under way and is now being directed toward the evolution of a scheme to treat high-grade ore and concentrate by cyanidation. Preliminary laboratory experiments have been sufficiently encouraging to warrant proceeding with tests on a larger scale, which are now under way.

Direct Cyanidation of Low-Grade Ores

The low-grade ore rejected at the washing and sorting plant is treated by the cyanide process as outlined in Fig. 9. The principal features are (1) a wet desulphurizing process which liberates silver from its

mineral combinations and renders it soluble in cyanide solution; (2) precipitation of silver with aluminium dust, and (3) the recovery of about 92 per cent of the silver from a 30-oz. ore, in the form of bullion 997 fine. Briefly, the process consists in grinding to slime in caustic soda solution; desulphurizing by contact with aluminium; filtering to remove caustic soda solution; treating with cyanide to dissolve silver; filtering and precipitating the solution with aluminium dust; melting and refining the precipitate.

The milling plant is a model of excellence in design, construction and arrangement, and shows the value of good preliminary engineering. The arrangement of machinery and the relation of the several departments show a freedom from complications which are so apparent in many poorly designed mills. Here even the casual visitor may quickly gain an idea of the system of treatment. All machines are readily accessible for inspection and repairs, and cleanliness is evident throughout. In short, it is apparent that Nipissing ideals are comprehensive and extend to every department of the work.

Extremely Fine Grinding Necessary

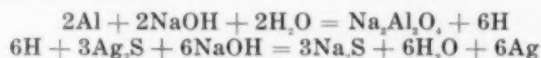
The ore is ground in 0.25 per cent. caustic soda solution, with 5 lb. of lime per ton of ore added at the battery bins to aid in slime settlement. The equipment for this part of the treatment consists of forty 1500-lb. stamps, four 6 ft. by 20 ft. tube mills and seven duplex Dorr classifiers. In order to secure a good extraction by cyaniding it is necessary to grind the ore extremely fine—practically all minus 200-mesh. For this reason the capacity of tube mills and classifiers is unusually large when compared with stamp capacity.

The battery discharge, 7 parts of solution to 1 of ore, is distributed to four of the Dorr classifiers. These machines have higher overflow weirs than usual to prevent the discharge of fine sand; they are also about 5 ft. longer than standard machines and have bevel-gear drives. The slime overflow passes to collecting tanks, and the sand discharge is delivered to the two coarse-grinding tube mills and ground with 40 per cent of solution. The tubes, in turn, discharge to three more Dorr classifiers, the slime overflow from which passes to collecting tanks, while the fine sand is delivered to the two fine-grinding tube mills which discharge back to the set of three Dorr classifiers. Thus the sand is kept in a closed grinding and classifying circuit until it is ground fine enough to overflow the weirs of the classifiers.

Desulphurizing with Aluminium

The ore ground in caustic soda solution is collected in tanks and allowed to settle. The excess solution is collected in overflow launders and through decanters until the remaining pulp consists of about 1½ parts solution and 1 of slime. It is then ready for the desulphurizing treatment, which decomposes the refractory minerals that contain part of the silver in the ore. Most of the silver is in the metallic state, but some is combined. Thus in a 30-oz. head sample, approximately 80 per cent of the value may be present as metallic silver and 20 per cent in mineral combinations.

The desulphurizing process is the result of research by J. J. Denny of the Nipissing staff.* His explanation of the decomposition of silver sulphide by the treatment with aluminium in alkali solution is represented in the following equations.



Similar reactions occur with sulphantimonides and

sulpharsenides. These reactions are brought about by grinding the thickened alkaline pulp in a 4 ft. by 25 ft. tube mill, lined with aluminium and carrying a load of about 4000 lb. of broken aluminium ingots. This is followed by mechanical agitation in vats lined with sheets of aluminium. At the end of 24 hours' agitation the pulp is filtered, discharged with about 26 per cent solution and treated by cyanidation.

Cyanide Treatment

Solution of silver is accomplished by 48 to 60-hr. agitation of a pulp consisting of 2 parts 0.25 per cent cyanide solution and 1 of dry slime. The agitation vats have mechanical stirrers and air lifts which are kept in constant operation. The air lift is placed near the periphery of the vat and has two discharge openings one above the other pointing in opposite directions. The combined effect of mechanical stirrer and air lift is a very thorough mixture and agitation of the pulp. After agitation the charge is allowed to settle. Clear solution is decanted and the remaining pulp is stored in tanks where it is kept in agitation until drawn to the filter.

Both alkali and cyanide filters are of the Butters stationary type and are equipped with the latest form of Butters vacuum filter leaf.[†] The cake is dislodged by the application of water from a pipe extending along the top on the outside of each leaf, and no reverse internal pressure is applied. The cyanide filter has 80 leaves and handles charges of 35 to 40 tons in about three hours. The cake is discharged with 26 per cent moisture. The adoption of the new filter leaf has resulted in a number of benefits. The cost of filtration is reduced, filters require less attention, and the fabric lasts longer. Ten-ounce duck is now used, which stands the acid treatment better than the lighter fabric formerly employed.

Precipitation by Aluminium Dust

Although zinc dust precipitation was a part of the process as originally planned, it was abandoned because experimental work, showed a rapid fouling of solution and loss of dissolving power. Further tests with aluminium dust finally led to the adoption of that precipitant with beneficial results. It was found that mill solution did not become foul, and that it was even more active than freshly prepared solution in dissolving silver. Other benefits obtained in practical operation are (1) the regeneration of 1 2/3 lb. cyanide per ton of ore treated, and (2) the recovery of a silver precipitate carrying about 27,000 oz. per ton. The subsequent melting and refining of this precipitate yields bullion 997 to 999 fine.

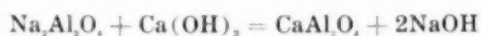
There is a tendency for soluble sulphides to form and accumulate in the barren solution. In order to oxidize these sulphides, the barren solution is aerated by means of air lifts before it is used again. His investigations on this point lead Mr. Denny to believe that soluble sulphides form as a result of the reduction of thiosulphates by aluminium in alkaline solution. If these sulphides were not oxidized in some manner they would tend to form potassium sulphocyanide, and this salt, Mr. Denny points out, is not convertible into soluble sulphide by aluminium and caustic soda except under conditions of concentration that do not obtain in mill solutions.

Aluminium does not accumulate in solution as some have surmised, but remains constant at about 0.08 per cent to 0.1 per cent, as in the case of zinc. Accumulation is prevented by precipitation of the aluminium as

**Min. & Sci. Press*, Sept. 27, 1913.

[†]This leaf was described in this journal Jan., 1914, p. 65, and a similar leaf was referred to in our issue for July, 1914, p. 472. Recent patents relating to Butters slime filtration are numbered 1,078,994, 1,096,132, 1,100,217-18-19-20-21-22-23-67.

calcium aluminate during the cyanide operation, and this passes out of the mill with the filter residue. It will be recalled that the battery pulp is charged with lime to aid in settlement, and that the residue from the dewatering filter still carries 26 per cent of the original alkaline solution with its quota of lime. When cyanide solution carrying some aluminium is added to this pulp the reaction between lime and aluminium occurs.



Cost of Producing Silver

In his annual report for 1913, Mr. R. B. Watson, general manager, gives the following interesting data on Nipissing operations. The low-grade mill was in operation 86.8 per cent of the possible running time, and treated 77,240 tons of ore. Stamp duty was 6.09 tons per stamp per day. The recovery of silver from the beginning of operations on Nov. 11, 1912, to Dec. 31, 1913, was 91.85 per cent. The ore-treatment cost, based on the tonnage given above, was \$4.132 per ton, distributed as follows:

	Cost per Ton
Crushing at mine.....	\$0.163
Aerial tramway.....	.100
Surface tramways.....	.117
Picking plant.....	.217
Crushing.....	.081
Battery.....	.267
Tube mills and classifiers.....	.547
Slime collecting.....	.448
Intermediate filtering.....	.127
Treatment.....	1.021
Filtering and discharging.....	.186
Clarifying and precipitation.....	.301
Refining.....	.127
Heating.....	.113
Water supply.....	.031
Construction.....	.286
	\$4.132

Consumption of Supplies at Low-Grade Mill

	Lbs. per Ton	Cost per Lb.	Total Cost per Ton Ore
Cyanide.....	5.190	\$0.1500	\$0.7785
Caustic soda.....	3.200	.0214	.0684
Lime.....	6.640	.0040	.0266
Aluminium dust.....	.568	.3373	.1917
Aluminium plates.....	.268	.2438	.0654
Aluminium ingots.....	.964	.2207	.2128
Flint pebbles.....	6.540	.0098	.0646
Ore pebbles.....	3.100	.0029	.0091
Coal for heating.....	30.160	.0029	.0881
Power, K. W. H.....	51.570	.0115	.5933
			\$2.0894

The cost of producing 4,552,173 oz. of silver in 1913, at both low- and high-grade mills, was 24.09 cents per oz., which is 6.7 cents higher than in 1912, when practically the same quantity of silver was produced, viz., 4,688,000 oz. The increased cost is due to the fact that nearly half the production in 1913 came from low-grade rock, whereas in 1912 practically the entire production was from high-grade ore. The cost of producing silver in 1913 is distributed as follows:

Cost of Producing Silver—Based on Production of 4,552,172.59 Ounces and 77,240 Tons Milled

	Per Ton Ore	Per Oz. Silver
Hydraulic mining and clearing land.....	\$0.554	\$0.0094
Diamond drilling.....	.093	.0016
Development and exploration.....	3.685	.0625
Stoping.....	2.191	.0372
Loading and sampling.....	.064	.0011
Assaying, engineering and research.....	.151	.0025
Administration and office.....	.381	.0065
Boarding house and camp maintenance.....	.278	.0047
Insurance and taxes.....	1.013	.0173
General and legal expense.....	.366	.0062
High-grade mill.....	.618	.0105
Low-grade mill.....	4.144	.0702
Custom Milling by Nova Scotia Co.....	.017	.0003
Depreciation.....	.793	.0135
Marketing product.....	.305	.0052
Corporation, New York Office and Travelling.....	.155	.0026
	\$14.808	\$0.2513
Less rents and interest.....	.615	.0104
Total operating cost.....	\$14.193	\$0.2409

Dominion Reduction Mill

This is a custom mill treating ore from the Kerr Lake, Crown Reserve, Cobalt Comet, Drummond Fraction and other mines in the district. The method of ore-treatment is outlined in Fig. 10. As the mill is run on a custom basis, few data of operation are available. One of the features of the dry crushing department is the recent installation of a 48-in. Symans horizontal reduction disc, the only one in the district.

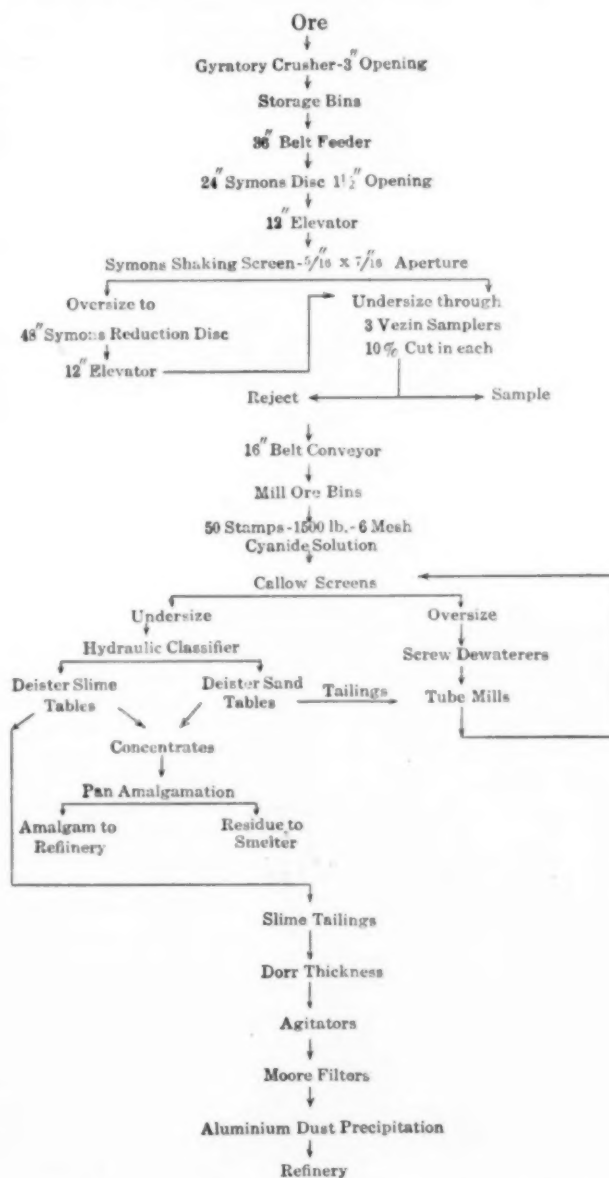


FIG. 10—FLOW SHEET, DOMINION REDUCTION MILL, COBALT

The ore is stamped in solution carrying 3 1/2 lb. cyanide and 2 1/2 lb. protective alkalinity. Sodium hydroxide and lime are used for the latter purpose, about one-third caustic soda and two-thirds lime. After sizing through Callow screens the pulp is classified for concentration, and sand tailings are ground in tube mills in closed circuit with the screens. Concentrates receive a special treatment by pan amalgamation in strong cyanide solution. The products are (1) amalgam, which is melted and refined, and (2) a residue containing about 150 oz. silver per ton, 33 per cent arsenic, 6 per cent cobalt and 3 per cent nickel. This residue is sold for further treatment.

There are no unusual features about the cyanidation of the slime tailing, except that aluminium dust is used as a precipitant of silver from the pregnant solution. The dust is fed at a constant rate of 3 lb. per hour when solution pumps are running, regardless of the grade of the solution. Mixing is accomplished in a small wooden tank, in the bottom of which runs a screw propeller that forms a vortex. Precipitation efficiency varies with the grade of solution and is more nearly complete when the solution carries about 10 oz. silver per ton. Thus from a 9-oz. head solution the tail solution carried only 0.12 oz., while from a 4-oz. head solution the tail solution would carry 0.82 oz.

In conclusion the writer desires to record his appreciation of the many courtesies extended during his visit to the mills in Cobalt, where every facility was afforded for gathering the notes on which these articles have been based.

The Late Walther Feld

The death of Walther Feld on March 15, this year, in Linz on the Rhine, Germany, has left a void in the chemical profession which will be very difficult to fill.

Feld was a genial investigator, an unexcelled chemist and a never-tiring pioneer in the field of chemical technology, which, combined with a loving personality, made him a friend much sought after.

Feld was best known through his origination of the barium industries in Germany, and a walk along the banks of the Rhine shows plainly to what extent his labors in this field bore fruit, as the principal industry for many of these villages depends upon the production of barium compounds. He created this industry when there was no market for the product, but due to his untiring efforts, this market was found for the benefit of the present producers.



THE LATE WALTHER FELD

Being called upon to investigate tars and oils in coal gases, he was led to establish a theory of fractionately separating various tar products in the gas through distillation by the own heat of the gas, and his results in this endeavor can be pointed to with pride in many of the European coke-oven works. Having once taken in hand the condensation of coal gas, he did not stop until he had solved the old problem of purification in closed vessels—something which gas men had been seeking for the last fifty years. Not only did he make it possible to purify in closed vessels, but he was also able to combine the hydrogen sulphide contained in the gas with its radical of ammonia, and thus produce sulphate of ammonia* without the use of sulphuric acid, and perhaps his only regret before joining those in the Great Beyond was the fact that one of the largest installations of his direct sulphate recovery system, and which had been built under his supervision, was to be

placed in operation ten days after his death. While this disappointment during the few days of his illness had to be borne, he still had the satisfaction of seeing the rosy dawn of success in a field which had been abandoned by many others.

While Walther Feld was not known as well in the United States as he was in Europe, his demise is of interest to all engaged in like pursuits, as he hoped that many of the chemical principles laid down by him would be followed by his brother chemists the world over.

In closing the notice of his death, nothing of greater praise can be said than the following words which appeared in *Die Chemische Industrie*, published in Germany, namely: "The barium industry as well as industries connected with tar and ammonia must thank him for the new paths which he has broken, and in the history of the development of German chemical industries the life work of Walther Feld must for all time remain an imperishable page of fame."

FRED. H. WAGNER.

Niagara Falls Meeting of American Electrochemical Society

For the fall meeting of the American Electrochemical Society, to be held in Niagara Falls from Thursday to Saturday, Oct. 1 to 3, the local committee has announced the following provisional program:

On Thursday morning there will be a session for the reading and discussion of papers, in the afternoon an excursion to Fosters' Flat, in the evening a smoker "with an unusual program."

On Friday morning a visit will be paid to plants and power houses, with a high-voltage display at one of the power houses. In the afternoon there will be a session for the reading and discussion of papers, and in the evening a dinner, followed by an illustrated lecture.

On Saturday morning and afternoon an experimental symposium will be held at the Fitz Gerald Laboratories, and in the evening a kommers at the Fort Schlosser Club.

The privilege of the local golf course will be extended to members. Other special features of the program will be announced later.

Salt Lake Meeting of American Institute of Mining Engineers

The Salt Lake City meeting of the American Institute of Mining Engineers will be held from Monday to Friday, May 10 to 14, 1914.

Technical sessions will be held in the evenings of Monday, Tuesday, Wednesday and the morning of Thursday in the ballroom of the Hotel Utah.

The program of excursions is very full and exceedingly interesting. Practically the whole of Tuesday will be devoted to a trip to the mines of Bingham, the concentrating mills of the Utah Copper Company at Mazna and Arthur, and the Garfield smelter.

On Wednesday the smelter of the American Smelting & Refining Company at Murray and the smelter of the United States Smelting, Refining & Mining Company at Midvale will be visited in the morning and the smelter of the International Smelting & Refining Company at Tooele in the afternoon.

A trip to Saltair Beach is scheduled for Thursday afternoon and an automobile trip to Big Cottonwood Canyon, with picnic luncheon, for Friday. On the evening of Friday there will be a banquet at the Hotel Utah.

*See his article in the July, 1912, issue of *Metallurgical and Chemical Engineering* (vol. X, p. 395).

Research Work in the Laboratory and Mill

BY WARREN F. BLEECKER

Technical men seem to be agreed that the time is not far distant when industrial concerns will realize very forcibly the necessity for greater efficiency in operations and processes of a chemical and metallurgical nature. Competitive conditions will probably require certain classes of manufacturers to produce not only better grades of materials, but also to produce them somewhat more cheaply than in the past.

We may also imagine a time, not far distant, when the people as a whole may caution owners of mines and mills not to waste by crude manufacturing methods substances which are at the same time useful and rare. New minerals will be found in workable quantities, and processes must be devised for them. Methods of treatment for lower-grade materials will be required from time to time. And finally it must come to pass that processes will be devised which will be dependent upon by-products from industries wholly diverse and unrelated.

These things are now being thought of more and more, and men of all grades of technical training and ingenuity are being called upon to attack the problems already discerned by farseeing directors of industrial organizations.

Research Ability

It has been observed that a great many managers of plants, chemists, and young technical graduates have some difficulty in approaching certain classes of experimental work which they are called upon to do. This is especially true when a man does not possess a natural aptitude for research work, or in case the problem to be solved involves a branch not directly within his experience. For example, the manager of a wet concentration mill may find it necessary to investigate the possibilities of other methods of treating his ore, such as magnetic separation, or cyanidation. Facing such a problem, he should feel assured that he has access to certain avenues of investigation which are wholly within the limits of his ability and experience. The fact of his proficiency along his present line of work should give him assurance of being able to carry to a successful issue such experiments as he may undertake.

It is a distinct economic loss for a man of average intelligence to avoid lines of experimental work, essential to his business, but which he assumes to be out of his immediate mental grasp, because they happen to involve special knowledge somewhat apart from his previous experience. While it may take a lifetime to become an authority on electrometallurgy, for example, a working knowledge of the principles of this art may be acquired almost by wishing it.

While recognizing the necessity for expert and thorough knowledge we should not overlook the advantage of a perspective which embraces accurate detail and at the same time mellow it.

Each man who is doing his work well has acquired a certain mental equipment which is adapted to his present needs. I wish to point out that this equipment, with the aid perhaps of the knowledge of a few fundamental principles, will serve equally well in such new problems as may be attempted. In a word, nothing within reason is impossible to the man with sufficient courage to tear apart the veil of mystery surrounding unfamiliar fields and confidently examine the various elements. Such a procedure will yield ample returns for the small effort invested.

It would be unnecessary to offer suggestions to trained investigators and presumptuous to advise men

expert in their particular lines. But of these I ask the privilege of stating facts broadly, because once the attempt is made to state exceptions and to define the innumerable phenomena secondary to a given principle, there is immediately set up a perplexity, however small, in the mind of the beginner, and the entire purpose is defeated.

I do not think it necessary in the discussion of electrolysis, for example, to send out a general alarm as to the theories of osmosis, ionic migration, polarization, decomposition voltage, equilibria, and kindred phenomena. The specialist understands the importance of these things from sources of unquestioned authenticity; when the other fellow gets to a point where he can quibble over definitions and theories, he will not have been far misled by getting a few facts in the rough.

To him who has had the privilege of receiving instruction from high-class industrial engineers of our colleges and schools, or to him who possesses a genius for experimental work, I have but little to offer. It is to that class of men who are called upon to perform tasks somewhat outside their training or experience that encouragement should be given, not by laying down rules and laws for an indefinite number of specific cases which might arise, nor again to destroy or distort individual methods or habits of thought, nor again by attempting to diagram every imaginable case which might arise, but by speaking of certain basic principles, indicating possible avenues of approach, by assuring them that many new things are actually within their experience, and finally by pointing out a few sources of error and fallacy which are certain to be encountered.

Scope of Metallurgical Research

Research may be undertaken with any one of a number of different objects in view. It may be desired to give an ore a preliminary treatment in order to obtain a product previously rejected; a demand may have arisen for the improvement of a certain product without an expensive and unnecessary improvement of other products; it may be desired to improve the grade of all products; some reagent used in the processes may have become more expensive or rare; a new and different ore body may demand treatment; a general shaking up of processes to the end of greater efficiency may be required; and there are hundreds of other reasons why researches are undertaken. In each case there will be found a logical chain of possible incidents between the requirement and the hoped-for result, and this chain ought to be diagrammatically stated for constant reference.

Statement of the Problem

One of the first things an experimenter should do is to state his problem as definitely as possible in writing. He should state the result hoped for, the allowable variations of present practices, allowable costs, grades of products, available raw materials, etc.

I have found it useful to imagine a process or operation completed, and then look back over it to see what further could be desired. There is an old saying as to the superiority of hindsight over foresight. Sometimes one may be credited with considerable foresight when it has been nothing more than hindsight. I have heard this expressed a little more elegantly by one of my professors in mathematics. He called it "Back Illumination." This back illumination leads us to be more truthful with ourselves, for the reason that a careful survey of that which we are supposed to have done will surely reveal defects in a system which were at first sight hidden by our natural optimism.

I do not recommend that one should attempt to

imagine all of the finished details at the outset. On the contrary, to determine details is the object of the research. But I insist that a plan of action should not be left to chance reactions and phenomena which may from time to time be observed. Before a series of experiments can be successfully attempted one or more properties or peculiarities must be known, which properties have the possibility of lending themselves either to or toward the end desired. When, as is often the case, advantageous properties are not known, a definite and thorough research must be made until they are found. It is not within the limits of my experience that this is impossible.

Optimism Versus Proof

Without going into the psychology of the thing, I am convinced that there is no other profession which exacts from its members such a rigid adherence to truthfulness and honesty between man and man. In no other profession that I know of does there exist such an amazing and constant series of opportunities for optimism and hopefulness to shade into realities without the formality of rigorous proof. It is altogether too easy to assure one's self that a little more heat, a little more reagent, or a little more time will result in a higher extraction or a greater degree of purity, without actually performing the operation. Among other sources of opportunity for self-deception may be mentioned the neglect of traces of impurities which will surely multiply in finished products; lack of appreciation of the time element; neglect of allowance for wear and tear of apparatus, and finally, the failure to put down in cold figures the losses at various stages of the treatment through which a product must pass.

In all cases the experienced metallurgist will base his final conclusions on recorded observations. It is a practice to be recommended to the investigator who is working along lines in which he is not expert.

The Importance of Qualitative Results

In the exploration of new fields, the importance of obtaining results which are only qualitative, can hardly be overestimated. The idea of the primary mass has even a more profound meaning in chemistry and metallurgy than in freehand drawing. If one will but take time to block out his problem he will obtain a perspective worth many times the effort. The technical research man must allow his train of thought and his habits a certain flexibility, in that he should be able to rush through almost countless operations, making them serve his purpose without the useless labor of completing each for the sake of art. In other words, he should be able to learn many lessons from unfinished work without jeopardizing his instinct of accuracy. I have found it a most difficult thing to train young technical men in the habit of knowing when an experiment is finished.

I will illustrate this point as follows: It is known that the values in certain radium-bearing ores may be partially separated by a system of concentration involving flotation in a fluid. Assuming that an experimenter in attacking this problem had determined with a gold pan, a microscope, and an electroscope, that the process had a commercial possibility, he might proceed to verify this in a number of ways, but I will suggest a simple experiment.

One hundred grams of ore pulverized to suitable fineness is agitated with water in a litre Florence flask. After allowing this to settle for a few minutes, the supernatant slime is decanted off, more water is added, and the operation repeated. Washing is continued without regard to the quantity of water until further agitation removes no slime. Assuming that water con-

centration is to be alone considered, it is obvious that under the conditions the maximum values were removed from the coarser sands. In other words, providing all of the slimes were to be saved and all of the tailings were to be rejected, the difference between the analysis of the original ore and the analysis of the tailings, expressed in percentage, would represent the highest possible efficiency.

In order to ascertain the maximum limit of recovery, therefore, it would only be necessary to know the original value of the ore, the weight and value of the tailings. In this particular instance the labor saved in rejecting the slimes would be several times that necessary to wash and dry the tailings, because of the difficulty in filtering and drying the slimes. It will be obvious that this operation will have to be repeated a large number of times on the various classes of ore under consideration, and for this reason, the saving of time will be a great factor in getting without delay a solid foundation for the refinements which must come later on.

The same rule will apply during the study of the quantities of water necessary for concentration; once it is established that a certain separation can be effected, it is entirely unnecessary to check the results simply because there is an opportunity. Time is so very valuable that it should not be wasted in doing unnecessary work.

Almost all results which are obtainable in a small way, especially through the application of laboratory methods, cannot be taken at face value. In other words, they must have the proper interpretation before reliable conclusions can be drawn. The experimenter must by experience familiarize himself with large-scale operations before he attempts to draw conclusions as to costs, extractions, recoveries, grades of products, and other operating data.

Laboratory Reactions

On the other hand, there are but few reactions peculiar to the laboratory. As a rule, mill operations are more nearly quantitative and may be obtained in a more satisfactory manner. Further than this there are certain classes of reactions which in the mill are almost marvels of simplicity and perfection but which are practically impossible in a small way. Among these may be mentioned magnetic roasting, salt roasting, gas washing, extraction by agitation, magnetic separation, wet concentration, and superheating.

Unfortunately there seems to be a widespread misunderstanding concerning the relation between laboratory work and mill operations. It is a very common belief that laboratory operations are very unreliable and even dangerous when considered as a basis of a proposed mill operation. That mistakes have been made and that a prejudice has been nourished therefrom, is due more to a lack of common sense than to inherent defects in small operations.

Chemical analyses do not, as a rule, give finished products, nor allow of simple multiplication in order to become of commercial application. While a standard method of tungsten assay results in the actual production of a minute quantity of tungstic acid, and the assay of gold results in the production of the pure metal, there are many other elements which are not determined by weighing small quantities of a finished, marketable product. Experience has taught us, however, that so many cubic centimeters of permanganate means a definite percentage of lime, and that a given quantity of standard cyanide tells us the percentage of copper. Because no further significance is attached to such reactions they have not fallen into bad repute by being introduced as mill processes.

One of the first things learned in chemistry is the group separation of certain of the elements. During that awful first year of the study of qualitative analysis the student picks up more misinformation than he can forget during the remainder of his life. While he is being taught methods he thinks he is learning principles. It is the application of these false principles which contributes so largely to the misunderstanding of the proper relation between experimental laboratory work and mill operation.

If one will only look upon laboratory experiments as sources of information and not as achievements; as indicative of detached facts rather than complete processes; as representing arguments, and not conclusions; we may then free ourselves from fundamental error in this matter. If we but realize that chemical reactions, like human actions, taken singly or in different settings, reveal tendencies rather than habits, the situation will be more sane, and our conclusions more worthy of our efforts.

It is often said: "Of course, you can do it in the laboratory, but in the mill it will be different." Nothing can be farther from the truth. In the entire realm of chemistry and metallurgy, I defy any man to mention as many as half a dozen operations which can be done only in the laboratory. A blast furnace is more efficient than a blowpipe, a jet of steam will heat a tank of liquor in the same time a beaker is heating over a burner, and with modern apparatus, tons of iron hydrate can be filtered, washed and dried as quickly as grams of the same material are likewise treated in the laboratory.

It is almost discouraging to find so many still speaking of the laboratory as a place where wonders are worked. I believe that it is possible to get at the bottom of such unscientific ideas and root them out forever.

It is, of course, a matter of common knowledge that the general conditions affecting chemical reactions are temperature, concentration, excess of reagents, presence of interfering elements, physical conditions of the reacting substances, and the tendencies of the various reactions to proceed at given rates. Among such a large number of conditions it follows that the number of combinations is very large. The lack of control of any one condition may mean lack of control of all; and there may be more than one condition during the inception of a process which is neither properly controlled nor understood.

In the treatment of small quantities of ore in the laboratory, there is always the danger of using a quantity of reagent entirely out of proportion to that which is allowable in a large way. Five cubic centimeters of sulphuric acid used to decompose a gram of ore containing 5 per cent copper means 182 tons of acid per ton of copper. Five grams of sodium carbonate necessary to decompose one gram of roscolite carrying 2 per cent vanadium pentoxide means 445 tons of soda ash per ton of metallic vanadium.

The proper place to start an investigation is the laboratory, however large the final scale of operation is to be. There are a great number of reasons for this procedure, and, so far as I know, there are no reasons against it.

The Process

It may be pointed out that the really important reactions of a new process are very few—usually not more than one. These reactions must be studied at close range in order to ascertain all favorable and unfavorable conditions. There will be numberless secondary reactions and phenomena, many of which will in themselves present problems. At best the thorough investi-

gation of unfamiliar reactions and the resulting products requires scores, and sometimes hundreds of experiments. All else aside, with only one lifetime at his disposal, one cannot afford to begin a comprehensive research on a large scale.

We may arbitrarily divide an ore treatment process into three parts: the dressing of the ore, the extraction of the values, and the recovery of the products.

It will be apparent that no one of these departments may be completely organized without due regard to at least one of the others. For example, certain phases of roasting, concentrating, or leaching may determine the limits, and in a large measure the manner, of grinding. It is equally true that the separation or extraction process will have to conform to the requirements of the final production. In other words, the entire process will, as a usual thing, have to be built up as a unit. If the manner of grinding is decided upon before the extraction process is fully worked out, the experimenter has seriously endangered his prospects, because of the limits within which he must now work.

It often happens that the first-discovered and most desirable reactions of a series of proposed processes for ore treatment occur at a more or less advanced stage. In such cases the metallurgist will build toward such reactions from the ore pile, and from the finished products as well. The salt roasting of vanadium ores, and the dry chlorination of sulphide ores illustrates this principle.

Magnitude of Experiments

It would be a serious mistake to confine experimental work to the laboratory until a series of processes are completely organized. Let it be supposed that a satisfactory extraction of a sulphide ore with chlorine could not be obtained in laboratory apparatus on account of lack of control of temperature, agitation, etc. It would, therefore, be necessary to use such apparatus as would give satisfactory results, although the ore for the experiments might have to be ground on a bucking board, and the resulting solutions treated in beakers. That is to say, because it is necessary to perform one of a series of operations on a large scale, I do not think it advisable to work on this scale entirely.

Archimedes, or Franklin, or Hubbard should have formulated a rule something like this: Get your information with as little expenditure of time, money, and material as possible, but get it.

A gentleman with the ink on his Doctor's degree hardly dry, approached a manufacturer with a claim that he could save a certain more or less valuable by-product at that time being wasted. That is to say, he had a process, admittedly in the embryonic stage, which sounded plausible to the manufacturer. After some discussion the chemist was employed and given permission to direct the perfection and installation of his process. Inasmuch as he had made no claim that the process had been tried or that it had been developed to the operative stage, it would have been the logical thing for him to have spent a few months or weeks as the case may have warranted in studying the proposed reactions and planning certain details. Instead of doing this, he ordered a quantity of special apparatus and began mill operation. Things went wrong here and there, and he started again and again until he had everybody disgusted and wishing him out of the way. Of course, he failed and lost what would have been a good job. As his proposition was afterward explained to me I am inclined to believe that it had merit, and that it was worthy of investigation. This young man could easily have made a showing if he had but realized the necessity for common sense. It does not require the intellect of a Newton nor the genius of an Edison

to get results. One must have a good store of common sense and the courage to work, work, and work. If the idea is a good one, the results will finally unfold in an amazingly simple manner.

Control of Quantities and Conditions

It is so easy to get rid of a small quantity of an objectionable impurity in the laboratory with a few drops of this or that reagent that the investigator may for the time pass over the possible cost or difficulties involved in a large way. It is also relatively easy to obtain a complete precipitation, or decomposition, or dehydration by adding a few drops of alkali, a few hundred milligrams of peroxide, or by adding a few calories of heat. Although it is not the usual thing, these factors may multiply geometrically when applied in the large way. This is not an uncommon thing in drying semi-colloidal precipitates; a cake twice as thick requires more than twice as long and much more heat for dehydration.

Another matter worthy of attention is that, as is well known, many reactions depend largely upon the strength of liquors, both as to valuable content and excess of reagents. For our purpose let us assume that the concentration of a solution as obtained from an extraction process is of the proper degree for precipitation. It is desirable to precipitate at or near the boiling temperature and this condition may be easily and quickly done by heating the solution over a flame. It will be obvious that while the concentration is not appreciably altered during the laboratory treatment, the method of heating in the mill will be different. More or less time may elapse between the extraction and precipitation in the mill as compared with the same operations in the laboratory, resulting in a loss of heat, and therefore requiring more time for heating. Although the solutions may be easily and cheaply heated by the introduction of live steam, dilution may occur to such an extent as to materially lessen the efficiency of the reactions.

Such obstacles are by no means fatal to processes, but they should be foreseen and plans for compensating for differences in concentration, etc., must be carefully worked out before one has the right to claim a successful issue.

Interpretation of Results

It is almost a truism to say that the interpretation of quantitative laboratory results is quite as important as the interpretation of other phenomena which may fall under observation. For example, the fact that a certain extraction was obtained in the concentrating experiment mentioned above does not mean that exactly such results will be obtained in practice. It means that under the conditions of washing, etc., the limit has been reached, and that somewhat lower recovery may be expected. A microscopical examination of the tailings may suggest a slightly modified treatment, providing the possible recovery is not satisfactory. On the other hand, a careful examination of the slimes may reveal physical properties which will lead to the investigation of other modifications. The experiment has confirmed the supposition that a mechanical separation is possible and it remains for countless other experiments to completely disentangle the knot which Nature has so cunningly woven.

In the study of ore extraction, it often occurs in a series of experiments that after a given treatment the percentage extracted is strikingly constant. This is very significant in many cases, and may mean that the values exist in more than one state of oxidation, or that the treatment leaves unaffected only that quantity which is differently mineralized.

As a rule, one may obtain results in a large way which are more nearly quantitative than those obtained in the laboratory. Conditions of temperature, concentration, agitation, acidity, volume, etc., may be kept more nearly constant, and readily within the required limit.

In working with a new ore it is a part of the metallurgist's business to assure himself that the sample at hand is reasonably fair and representative; otherwise he may work serious injury to himself and his employer. The whole, if practicable, should be thoroughly mixed and sampled according to standard methods, and in almost all cases, a complete analysis should be made.

Some Valuable Accessories

Assuming that ore treatment is under discussion, it is a very natural thing to make a preliminary survey with the aid of a few simple accessories before undertaking a profound research. For example, the following equipment is indispensable:

- A complete set of screens.
- A rough balance, 1 gram to 1 kg.
- A gold-washing pan.
- A horseshoe magnet.
- A powerful reading glass.
- A microscope with micrometer slide.

With the screens, the experimenter will be able to determine among other things whether the vaules are evenly distributed throughout the gangue, or segregated to a greater or less degree in portions having different hardness, or other physical property affecting the size. A screen test may also indicate the proper fineness of crushing and at the same time indicate possible losses through dust and slimes.

A gold-washing pan may reveal the presence of elements which were not suspected, and will sometimes bring to light minerals which are present in quantities so small as to escape detection in the process of chemical examination.

A most valuable aid to the metallurgist is the microscope. With it one may not only study the size, form, structure and other characteristics of ore particles, but also observe certain classes of chemical reactions. With a few milligrams of ore on a glass slide one may study the action of such solvents as may be thought worthy of trial, and the resulting solutions may be further examined under moderately high power in investigating certain phases of precipitation or purification. As the dynamic test of metals discloses properties not even suggested by static tests, so the microscope will reveal facts beyond the ken of the analyst. For example, a very excellent and certain method of controlling the roasting of telluride ores was devised by Mr. William H. Davis in Boulder County. He suspected that the diminishing sulphur content of the ore as roasting proceeded was not a true index of the decomposition of the telluride mineral. Accordingly he took samples at various stages of the roasting process and separated the gold-bearing mineral from the gangue by washing in a gold pan. An examination of the minute particles from the various samples under the microscope at once showed the physical and chemical conditions resulting from under-roasting, properly roasting, and over-roasting. Mr. Davis not only proved his point, but succeeded in obtaining an extraction much higher than when using only chemical control.

A Study of the Ore

In the study of ore extraction it is always desirable to know, if possible, the manner in which the values are combined, both chemically and physically. With such information it is usually not difficult to attack the mineral directly, and in some cases it is quite possible

to leave the present gangue matter wholly unattacked.

In some classes of ores it is also necessary to know the potential reduction. This together with a knowledge of the analysis, will usually throw much light upon the existing conditions of the values, as to whether they are partially reduced or partially oxidized. An ore may be examined as to its state of oxidation in several ways, such as weighing a portion which has been previously ignited under such conditions as will remove the combined water, organic matter, etc., and subsequently oxidizing and noting the increase in weight. In an experiment with ore containing sulphur or certain other elements, it will obviously require various and ingenious methods to avoid compensating errors, but there is usually a way to do it. A partial solution is some neutral reagent, such as sulphuric acid, and direct titration with standard permanganate or some other agent will also lead to information.

Value of Approximations

The research engineer should accustom himself to grasp the meaning of percentages rapidly, and should be able to make such rough mental calculations as will at times allow quick work. Refined calculations are of great importance and have their place, but rough and ready approximations will save valuable time and material. A set of lead chambers could be seriously damaged if the man in charge stops to figure when he sees a storm coming up, or if he finds that the nitre man has forgotten his charge an hour ago.

Together with this faculty of rapid approximation should go a keen sense of order of magnitude. This is a most useful possession and will keep one from many errors, plausible enough on the face, but easily recognized by their very absurdities. For example, men who have a properly trained sense of the order of magnitude are almost free from errors resulting from a misplaced decimal point, especially in slide-rule calculations. The technologist should have such an intuitive sense of the fitness of things that will cause him to question at once results, sizes and forms which are not in keeping with accuracy or good practise.

Often in working with new products which are not of enough importance to warrant the development of routine analysis, the analyst may report results which seem to him to be perfectly reasonable, but which to the metallurgist are absurd. Perhaps I may have been unfortunate in meeting men whose ideas of the fitness of things were limited, but my experiences have convinced me that the problem is real.

The Order of Magnitude

A gentleman with several letters after his name suggested to me in all seriousness the catching of rain-water in an open tank for the purpose of supplying a plant with distilled water. That a year's rainfall would not have lasted an hour did not appeal to him in the least. The story of the engineering student who designed a brace one one-hundredth of an inch in diameter is surely not exaggerated. I knew an engineer who attempted to stop a hole threaded for $\frac{1}{2}$ -in. pipe by simply shoving in a short $\frac{1}{2}$ -in. bolt. He explained that he couldn't find a $\frac{1}{2}$ -in. plug and that the bolt would keep in some of the water anyway.

A certain large engineering concern was permitted to do the engineering on an electrode-regulating device. They were told that the electrode and holder would not weigh more than 100 lb. They sent a mechanism guaranteed to lift something like 9000 lb. Another engineering firm in or about New York City sent out a generator with a rated amperage of 2650, while the copper busbars from the brush holders had a section of

less than half a square inch. In addition to this, the connections between the brush holders and these bars were of cast iron.

We see advertisements of washing machines which will do the family washing for 2 cents, and of an apparatus which will furnish distilled water at less than 3 cents per ton. I think the whole matter is worthy of serious consideration.

The experimenter should be constantly on the lookout for unusual reactions and phenomena, and be prepared to take advantage of them, should they be suited to his purpose. It is usually upon reactions somewhat out of the ordinary that new processes are based. In other words, the most successful technologist does not always think of copper in terms of a hydrogen sulphide precipitate.

In the investigation of a complete process of ore treatment the experimenter is trying to determine two things: the efficiency of the process, and the cost. To a large extent these two things are interdependent, although the first question is answered only by experiment, and the second may be answered largely by calculation in many cases.

It must be remembered that the cost per unit of value recovered is finally going to be the point upon which the success or failure of the process is determined. A low efficiency at low cost is usually more likely to prove profitable than a high efficiency at high cost. The experimenter must at all times keep in mind certain limits beyond which he cannot go in the use of expensive chemicals, quantities of chemicals used, and the number of operations through which the ore and products must pass.

Choice of Reagents

Excepting for the purpose of obtaining information of a strictly chemical significance, it is usually not advisable to make use of reagents which are known to be inapplicable to mill use. This, of course, depends wholly upon the nature of the problem, and the manner of attacking it. There is a tendency with all of us to ward the use of such powerful reagents as nitric acid, sodium peroxide, potassium chlorate, metallic zinc, concentrated sulphuric acid, and the like. These things do have limited uses in large operations, but one should have a thorough appreciation of quantities, costs, and difficulties of handling such chemicals before attempting to use them in works operation. A good example of the use of an expensive reagent in the treatment of very low-grade ores is found in the cyanide process. This process, however, is a law unto itself and its like is hard to find. The use of mercury for amalgamation, and the use of platinum in the contact process are other examples, illustrating the absurdity of formulating a rule to be followed blindly. It may be said, however, that the use of expensive reagents must be accompanied with more care and expert attention than the use of such materials as common salt or sulphuric acid.

There are many instances in laboratory experimentation in which an expensive reagent may be used as a matter of convenience and saving of time. For example, it is sometimes desirable to oxidize solutions in order to obtain favorable conditions for precipitation or filtration. In the laboratory the easiest procedure is to add hydrogen peroxide. Quantitative experiments and subsequent calculation will usually show that for a given oxygen content sodium peroxide is cheaper, liquid chlorine is cheaper still, and finally electrolysis is the cheapest of all. Under some conditions air may be used for oxidation of solutions, and besides its cheapness it has other very desirable properties. The experimenter is entirely justified in using peroxide of hydrogen for his small experiments, providing he is

certain it has no other effect than the addition of oxygen and that the method finally to be employed is likewise applicable.

Cyclic Processes

In general, the experimenter should be cautious in the contemplation of cyclic processes; that is, those which involve the recovery of reagents over and over again. The chief source of danger in processes of this nature is the accumulation of impurities which finally render the solution inactive. An example of a successful cyclic process is the chamber process for making sulphuric acid. Although the conditions are highly favorable for the recovery of nitric acid and its subsequent reuse, it is significant that the nitre pots are very important adjuncts to the plant.

There was proposed not long ago a process for the treatment of ores with nitric acid. The nitric acid solutions containing nitrates were to be treated with sulphuric acid, and the nitric acid recovered in absorption chambers such as are used in the processes for the fixation of nitrogen. The difficulties to be encountered at the various steps of the process would probably be very great, but this is not all. The designer of the process evidently overlooked the fact that nitric acid owes its activity to the ease with which the NO_3 ions decompose. Thus the thing is defeated even before it has begun.

Perhaps the most satisfactory cyclic process of large magnitude is the cyanide process. The history of the development of this process shows that the early investigators met with their share of trouble, and but for the fact that it was so revolutionary and so necessary, it is altogether possible that it might long ago have been relegated to the metallurgical graveyard.

The continued reuse of solutions in processes for the extraction of copper has been tried repeatedly, and in some instances with a measure of success.

Under certain favorable conditions it is possible to recover chemicals with profit. As a rule, however, it is more desirable to adopt counter-current systems of leaching and washing in order to get the full value of the reagents added, and finally discard spent and barren solutions together with whatever impurities they may carry. Spent solutions of acid and alkali are not often recovered with profit.

Electrometallurgy

It is generally conceded that the application of electrical energy to metallurgical operations is increasing geometrically. Representing as it does a form of energy powerful yet easily controlled, ready for work at the throw of a switch, it is all the more remarkable that still greater use is not made of it. Rapid as the advance of electrometallurgy has been, it deserves to come into greater use. To my mind, the greatest retarding influence is the lack of full appreciation of the possibilities of the application of electrical energy to ores and ore products.

To the research man, unfamiliar with the commercial application of electrical energy, it may be said that the experimental difficulties in this branch of work are no greater than in any other. As in any other branch of science, there are certain rules and laws to be understood, but they are of interest and wholly logical. After all, common sense is nine-tenths of the electrometallurgical game.

Aside from power and lighting purposes, electricity is employed in metallurgy for two general purposes: heating and electrolysis.

Electric current may be supplied either as alternating or direct, the greatest difference between these two forms being that in the one case the current is rapidly

reversed throughout the circuit, and in the other the flow is constantly in the same direction. The very rapid reversing of a current gives rise to effects not met with in direct current, but which for our present purpose need not be discussed.

While either alternating or direct current may be used for heating purposes, only direct current can be used for electrolysis since in this latter phenomenon a continuous removal of electric charges is brought about at opposite electrodes. It may be considered that negative charges are given up at the anode and positive charges given up at the cathode. Obviously, electrodes connected to a source of alternating current become consecutively positive and negative in such rapid order that the electrolytic effect of one instant is destroyed or counteracted during the next.

In the electrolytic cell or electric furnace it is desired to obtain all of the energy potentially in some form of matter: a pound of aluminium, a pound of silicon, a pound of zinc, each represents a definite number of kilowatt-hours of electrical energy.

Electrolysis

An impression which is more prevalent than is generally supposed, is that the electric current by virtue of its passage through a solution or other medium, exerts force upon it in such a manner as to cause chemical reactions to take place regardless of any strictly chemical habit to the contrary. In other words, it is wrongfully assumed that the electric current may set aside chemical laws and perform various chemical acts wholly impossible without its aid.

We may assume that the electric current creates no chemical change in the interior of an electrolyte and be entirely correct. Any effect of the electric current which concerns the technical investigator is due to the removal of electric charges from chemical elements or groups of elements (ions) at the electrodes, and at no other place. What the current really does is to change the electrical conditions of compounds which are in contact with the electrodes and thus allow the element so electrically discharged a certain chemical freedom. The exercise of this freedom is sometimes mistaken as a phenomenon of the current itself.

If we will allow ourselves to look at this thing from a common sense point of view, we may for a time place to one side the mass of theories and highly specialized information (which indeed we shall find of great value later in our experience) and observe the action of electrolysis as a thing of extreme simplicity. What the practical man wants is results. If he has a sound working knowledge of electrolytic phenomena, he may not yet know all that is possible, but he certainly will know that some things are impossible.

Electrolytic Decomposition

The following illustration has been chosen as typical of electrolytic decomposition:

Sylvanite, a telluride of gold and silver, may be decomposed in an electrolytic cell. It is only necessary to place the crystals in contact with the cathode of a cell, the electrolyte in which is sodium hydrate. The tellurium is rapidly dissolved, while the gold and silver settle to the bottom as a black powder. The electric current releases metallic sodium on the surface of the crystal and from this point on the action is strictly chemical, and would take place if the current were discontinued. If the tellurium compound were placed at any portion of the solution not in contact with an electrode it would remain unchanged, however much electrical energy might be passing through the cell. If the electrolyte were sodium chloride and the sylvanite were placed on the anode decomposition would result

from the action of the chlorine deposited upon it. In this case the gold would go into solution.

A Maximum Chemical Concentration

The electrolytic cell provides a very efficient means of obtaining a maximum concentration of elements or groups in a state of freedom. For example, by the electrolysis of a salt of sodium we may obtain a decomposition at the cathode such as would be possible with sodium element, because, indeed, we actually plate metallic sodium on the cathode and any conductor of the first class which is in contact with it. Likewise we may get the action of pure chlorine by electrolysis of a chloride. The products of electrolysis are formed in relatively small quantities, and in many cases instantly reunite, but they continue to be formed as long as the electrolyte holds out and the current continues to pass.

Secondary Reactions

When common salt is electrolyzed, the two products are sodium and chlorine. The chlorine is partly dissolved in the solution and partly evolved in the gaseous state. The metallic sodium is immediately set upon by the water present and sodium hydrate results. The chlorine which is dissolved reacts upon the sodium hydrate to form hypochlorite, and other reactions occur, depending upon conditions of temperature, concentration, etc. It is obvious, however, that the formation of caustic or hypochlorite results from purely chemical causes, and would take place if metallic sodium were placed in water, or chlorine were led into a solution of lye.

Electrode Phenomena

If the electrolyte be sodium sulphate there will be plated at the respective electrodes, sodium and the group SO_4 . Since neither of these can exist alone in aqueous solution, a decomposition of water takes place with the evolution of hydrogen at the cathode and oxygen at the anode. It will be perfectly apparent that if the anode is of such a material as will form a soluble sulphate, it will be attacked and dissolved. It will also be apparent that by proper selection of electrolyte and electrodes the full effect of either the hydrogen or the oxygen may be obtained for reduction or oxidation respectively. For chemical and mechanical reasons, the size and shape of the electrodes have to be considered, and to obtain the best results, the electrodes must be proportioned according to the current available and the effect desired. Either oxidation or reduction may be obtained by properly regulating the relative size of the electrodes.

In the greater number of electrolytic operations it is desirable to use an anode which is not attacked. Unfortunately our choice is limited to platinum, graphite, fused iron oxide, lead or gold in some cases, and possibly a few others. In case the cathode is metallic, it will not be attacked. In case the cathode is an oxide it will either be reduced in situ or dissolved and finally find its way to the anode. As an illustration showing how both of these effects may take place simultaneously, a plate of copper vanadate (a compound of copper and vanadium oxides) employed as a cathode will become a plate of sponge copper, while the vanadium pentoxide will be dissolved and tend to be plated out on the anode.

There are many desirable reactions which may be obtained by using a soluble anode; that is, one which is attacked by the negative group or element plated out upon it. For example, vanadate of lead may be formed by electrolyzing a solution of sodium vanadate and using a metallic lead anode. Likewise tungstate of iron may be made by using an electrolyte of sodium tungstate with an iron anode.

The electric current will of itself neither oxidize nor reduce, neither decompose (except the electrolyte itself) nor synthesize; but it will give freedom to elements and groups of elements which in turn will do these things. Such is electrolysis.

Compounds may be broken up by virtue of their elements having been plated out at the electrodes, and these elements may reunite to form either the same or different compounds. The formation of such salts would take place at the moment, however, if the current should be discontinued.

Electrolytic cells may be divided into two general classes: those in which the two electrodes are in the same compartment, and those in which the electrodes are in separate compartments.

In the simple cell there may be obtained such reactions as electroplating, refining of metals, production of metallic salts by anodic decomposition, oxidation, reduction, etc. In the compartment cell, of which there are several types, there may be obtained practically all electrode products, the cathode material being kept separate from the anode material. Perhaps the largest industrial application of the compartment cell is in the manufacture of caustic soda and chlorine.

Compartment Cells

The most widely used type of compartment cell is the diaphragm cell, in which the electrodes are separated by a porous partition. This partition serves to prevent mechanical mixing, but allows the passage of positive and negative elements and groups which finally reach the electrodes and become electrically discharged and free. Depending upon its porosity, the diaphragm will also allow a certain amount of diffusion and the resulting contamination of the compartment product. The investigator may make a diaphragm of filter paper, canvas, wood, clay, asbestos, alundum, silica, or in fact of any porous non-conductor. For very small experiments, porous cups may be used. At one time I used large flower pots with considerable success.

Another type of compartment cell is one in which the electrodes are separated by glass or other insulating material, the partition extending into a bath of mercury. The mercury may be connected directly as a cathode, or in certain rare instances it may serve as a pair of electrodes. In this case the electrodes which are connected to the source of energy are suspended above the mercury, one on each side of the partition, the mercury in the positive side becoming negative, and in the negative compartment becoming positive.

A third type of compartment cell is one in which the electrodes are separated by a partition which extends below them, but which does not touch the bottom of the cell. Gaseous products, such as oxygen and hydrogen may in this manner be kept separate, or one gaseous product may be removed and a product from the other electrode allowed to remain, either in solution, or as a precipitate.

It will be obvious that the principle of electrolysis mentioned above holds for all classes of cells. Different designs are only for purposes of convenience, economy, and for taking advantage of the fact that at a given moment two components of a compound are chemically free and separated from each other mechanically.

After one has become familiar with the general principles of electrolysis and has personally observed the phenomena of such cells as he may construct, the various texts on the subject will be read with great interest and profit. To the investigator who is not of a mathematical turn, it is suggested that he get what he can out of these books which come under his observation in which mathematical expressions are frequent. It is a very rare thing indeed that an author succeeds in

concealing all of his thoughts from the fellow who is after facts.

The Electric Furnace

In the broadest sense, the electric furnace is an apparatus for the transformation of electrical energy into heat and applying this heat directly as desired. The heat is always obtained by the introduction of resistance to the passage of the current. This resistance is always in the furnace itself and may be obtained in a large number of ways, depending upon the effect desired.

No experimenter who may have occasion to attempt electric furnace investigation should be without a good descriptive text such as Stansfield or Blount.

The construction of an electric furnace is a very simple matter. The first experiments will usually be of a qualitative nature, and experience will indicate changes after each furnace operation which will require different shapes and sizes of crucible, different placing of electrodes, and different manner of lining. Build the furnace as easily and quickly as possible at first and do not worry over losing a little heat. There is a special economic law allowing quite large losses in experimental work.

Qualitative Experiments

It is a waste of time in the beginning to calculate and weigh slag components and reducing agents too closely. Since the dust and gas losses, amount of lining chewed up by the slag, the action of the bath on the electrodes, etc., are not known, more can be learned by a few trial heats than by much calculation.

It is also a waste of time to worry over the chemical constitution of compounds which are to be smelted. It is a safe rule to follow that the electric furnace will reduce the various oxides to a common denominator in a short time. The thing to know is the oxygen content, the base metal content, and about what sort of slag the impurities will form. It makes no difference whether the iron oxide is combined with the alumina, the silica, or both, so long as the ultimate analysis is known.

Interpretation of Results

Accurate notes should be kept of all important matters, such as quantities, time, instrument readings, etc., but they usually have no direct meaning. There is almost nothing in the first stages of any kind of experimental work which does not have to be interpreted. The fact that one uses two pounds of electrode for each ounce of metal produced has but little bearing on the problem as a whole. In technical journals and society transactions we sometimes read very painstaking reports of experiments, giving electrical efficiencies, recoveries, electrode losses per pound (or per gram), and in fact drawing supposed conclusions of commercial practice from the data per se. The results of the small experiment are of value only if understood.

Power Supply and Apparatus

Some who may wish to attack a problem by using electrical energy, may be held back by the fact of having no available supply of power suited to their requirements. To these I suggest a conference with the local manager of the power company, and failing to get him interested, a meeting with the head officials will get results. It is known that experiments with electric furnaces and electrolytic cells lead to the consumption of power, and I have never met with a power company which would not meet me more than half way in developing new demands for their product.

One may borrow old transformers, or rent a second-hand generator. I once bought an old transformer core

for \$4.90; an additional \$20 paid for the copper windings and small incidentals, so that the complete transformer cost about \$25. I could obtain as high as 30 kw for an hour or so and 20 kw continuously.

An electric furnace may with economy replace gas or coke furnaces in many instances. For example, a cyanide mill may be somewhat remote from the railroad, but may have electric power. Instead of hauling gasoline for refining, it would be a very simple matter to construct a small electric furnace holding a No. 60 crucible, and the cost of operation would be very small. In fact, the losses attendant upon working around the usual coke or gasoline furnace would more than offset any difference in cost. When using a coke-fired furnace the gold is refined by a man who has to protect himself from the heat with an asbestos face mask and a suitable shield for his body. One may work with an electric furnace at 1200 deg. C. with bare hands.

I believe an electric furnace will be devised which will enable zinc smelters to redistill blue powder and dross much more efficiently than in the retort. It has been recently said that the trouble with electric zinc smelting processes usually was that the inventor endeavored to cause the zinc vapor to go in one direction and the gases of combustion in another. The writer of the article was of the opinion that this was an obvious impossibility. I am not so sure about it; in fact I rather expect that furnaces will be devised which will do this very thing.

The Economic Phase

While attacking a problem of ore treatment with purely mechanical means, hydrometallurgical methods, electrolysis, or smelting (electric or otherwise), the economic phase must not be lost sight of. Unfortunately, no rule can be formulated giving limits between which a profitable cost of treatment might lie. This is because there are very few, if any, standard methods of treatment which will deliver a metal or other product at a cost which bears a fixed relation to the selling price.

There are, however, certain factors which may be calculated with some accuracy in each specific case, which when given their approximate cost value, will permit of a reasonable deduction being made from the gross value of the ore. These factors, among which are cost of mining, crushing, interest, depreciation, maintenance of laboratory, and allowance for actual recovery (60 per cent, 70 per cent, 80 per cent, as the case may be), can be determined with reasonable accuracy by reference to published data on costs of standard operations.

It will then appear that an ore which has a gross value of \$20 per ton will have a value of somewhat less after making a fair deduction representing costs up to the process proper as suggested above. There will also be a shrinkage of values on the other end, such as transportation, penalties, smelting charges, etc., all of which lie between the material as produced and its market value in a finished condition.

Between the limits imposed by the shrinkages as set forth there lies a domain of values which is affected by the process itself. Here may be found the profits if the process is not too expensive.

As a corollary it may be said that the more standard operations one can employ in devising a process, the more accurate will be his cost estimates.

In conclusion, I wish to call attention to the fact that I have tried to deal solely with generalities, using specific instances only as illustrations and not in any sense as descriptions. I have endeavored to state facts and opinions in such a manner as not to give offense to

specialists, and although there is an obvious appeal to research men for discussion and other angles on experimental work, there is implied no challenge.

I owe much to Mr. Arnold J. Guerber, chief chemist of the Standard Chemical Company, who has made many criticisms and suggestions.

Canonsburg, Pa.

Electrolysis of Fused Materials Containing Zinc

"The Electrolysis of Fused Material Containing Zinc" is the title of a paper by Mr. Chun Hao Wang presented as a thesis for the degree of Doctor of Philosophy at Columbia University. The investigation was carried out at the Columbia School of Mines under the direction of Professor Edward F. Kern.

The first part of the paper is a review of existing literature on the electrolysis of aqueous zinc salt solutions, the precipitation of zinc from fuse baths, and the electrothermal treatment of zinc ores. Mr. Wang points out that before the electrolysis of fused zinc electrolytes can be carried on to advantage, there must be found a salt or a mixture of salts which is fusible at a temperature below the boiling point of zinc ($920^{\circ}\text{C}.$), which is a good solvent for zinc sulphide or zinc oxide, and which will not volatilize at the temperature at which the electrolysis must be conducted.

In order to obtain these data the following experiments were conducted by the author:

1. Determination of the freezing points of several salts and mixture of the same.
2. Determination of the solubility of zinc sulphide and of zinc oxide in the salts, and in the mixtures, whose freezing points were determined, and the freezing point of the resulting fused solution.
3. Electrolysis of the fused solutions of zinc sulphide and of zinc oxide, finally selecting that bath which gives the most promise for the recovery of zinc from zinc sulphide, or from zinc oxide, by electrolysis.

The results of the experimental works on freezing points and solubility may be summed up as follows:

Sodium hydroxide freezes at $312^{\circ}\text{C}.$ The addition of sodium tri-sulphide (freezing point $650^{\circ}\text{C}.$) up to 35 per cent causes a rapid rise of the freezing point. Above 35 per cent of sodium tri-sulphide the mixture solidifies in two layers.

Sodium tri-sulphide and mixtures of sodium tri-sulphide and hydroxide, are good solvents for zinc sulphide and zinc oxide (2 to 7 per cent zinc).

Calcium chloride freezes at $744^{\circ}\text{C}.$ The addition of sodium hydroxide causes a reduction of the freezing point, which is intermediate of the freezing points of the components of the mixture.

Zinc oxide and zinc sulphide are soluble in calcium chloride and in mixtures of calcium chloride and sodium hydroxide only to a small extent (below 0.7 per cent zinc).

Sodium thiosulphate freezes at $804^{\circ}\text{C}.$, dissolving over 1 per cent zinc either as zinc sulphide or zinc oxide.

Sodium silicates: Sodium bisilicate freezes at $1002^{\circ}\text{C}.$ The increase of base up to 57 per cent, corresponding to sodium sesquisilicate, causes a decrease in the freezing point to $950^{\circ}\text{C}.$ The further increase of the base up to 67 per cent, corresponding to sodium monosilicate, raises the freezing point. Sodium monosilicate solidifies at $1025^{\circ}\text{C}.$

Undercooling occurs with sodium silicates which have a silicate degree varying between the bisilicate and monosilicate. Undercooling is prevented by stirring the cooling mass.

The formation temperature of sesqui-sodium-silicate

from a mixture of three molecular equivalents of sodium carbonate, and one molecular equivalent of silica is about $1200^{\circ}\text{C}.$, whereas the freezing point of this silicate was found to be $800^{\circ}\text{C}.$

Boric anhydride in certain proportions causes the reduction of the freezing points of sodium silicates. An excess, in the case of the silicates containing more than 40 per cent of silica, causes the mixture to become mushy. The addition of zinc oxide to sodium silicates containing about 25 per cent boric anhydride raises the freezing point of the mixture.

Calcium fluoride lowers the freezing point of sodium silicates, producing a thinner fusion. The addition of zinc oxide to these mixtures containing 10 to 22 per cent calcium fluoride hastens the solidification of the resulting mixture.

The author's experiments on the recovery of zinc by electrolysis of fused solutions may be summed up as follows:

The zinc which was liberated by electrolysis is of fused sodium silicate, and sodium tri-sulphide baths collected as pellets, which remained in suspension in the fused electrolyte.

A bath of sodium bisilicate containing 25 per cent of boric anhydride, 22 per cent of calcium fluoride and zinc oxide (2.3 per cent zinc) after electrolysis showed a loss in the weight of the zinc bottom which was added at the start.

A bath of sodium monosilicate containing 20 per cent of boric anhydride and saturated with zinc oxide (5.8 per cent of zinc) showed great corrosion at the anode. In one case 32 grams of deposited zinc were collected by the metallic bottom (current efficiency 14.7 per cent), while in the other runs there was neither loss nor gain in weight of the bottom. Zinc flame was liberated around the cathode surface. The zinc was found in the cold solid bath as disseminated pellets.

Reversing the current for a short period previous to the electrolysis was found to be beneficial in electrolyzing fused electrolytes containing zinc.

The electrical conductivity of the fused sodium silicate baths containing boric anhydride increases with the decrease of the silicate of the mixture.

Fused baths composed of sodium hydroxide and sodium tri-sulphide containing zinc sulphide have higher electric conductivity than the fused sodium silicate electrolytes containing zinc oxide.

Sodium tri-sulphide baths containing zinc sulphide (5.8 per cent zinc) gave a very thick layer at the bottom which, however, was rendered fluid by adding lumps of sulphur.

The zinc liberated from fused baths of either sodium silicate or sodium tri-sulphide, saturated with zinc, does not collect as a mass on the cathode, but as pellets, which are mechanically carried into the fused electrolyte and held in suspension. This was also found by Thomas and others to take place in their fused zinc baths, which they attributed to be due to the formation of colloidal zinc, or zinc mist at the cathode. This, however, may be due to the high surface tension of zinc at temperatures near its boiling point.

The low efficiency of electrolysis of fused baths containing zinc is due to the zinc forming as pellets on the cathode, which are carried back to the anode and there oxidized and redissolved. Another cause of low efficiency is the loss of zinc partly by volatilization, due to the high current density at the cathode, the temperature at the surface of the cathode being higher than that of the bath.

By surrounding the cathode by means of a refractory condenser, the electrolysis of fused zinc electrolytes may be conducted at temperatures above the boiling point of zinc, collecting the deposited zinc in the condenser.

The Determination of Phosphorus in Phosphor-Bronze

BY E. W. HAGMAIER

Phosphor-bronze is one alloy of which the chemist cannot say much as to phosphorus content until he has obtained the precipitate of phosphorus. Alloys called phosphor-bronze may contain anywhere from zero to 1 per cent or more of phosphorus. An experienced analyst may get somewhat of an idea as to whether the phosphorus is high or not by observing the condition of the metastannic acid in dilute nitric solution. If the phosphorus is $\frac{1}{2}$ per cent or so, the metastannic acid will keep the original form of the drillings, after the copper, lead, etc., have gone into solution. The metastannic acid will be a grayish black when on the filter paper.

The writer has had occasion to analyze numerous phosphorus alloys in the last year and a half, and while doing so worked out an accurate and rapid method which will follow after a brief review of some of the older methods.

One method which is used a great deal is as follows: Dissolve one gram of the alloy in 30 c.c. of (1-1) nitric acid, and evaporate to a paste. Take up with a little dilute nitric and hot water, and boil to dissolve the copper, lead, etc., which has separated out. Remove from the hot plate and filter. The phosphorus is with the metastannic acid, provided there is sufficient tin present. The writer's experience has been that 6 per cent to 8 per cent of tin will hold up over 1 per cent of phosphorus. This precipitate is ignited in a tarred porcelain crucible, and the weight of the combined oxides of tin and phosphorus recorded. Potassium cyanide is then added to the crucible, and a fusion made. This converts the tin to metallic tin, and the phosphorus to potassium phosphate. The melt is then leached out with water, and the tin filtered off. The filtrate is made acid with nitric acid and boiled to drive off the cyanide. After all is driven off filter off any silica which may have come from the fusion attacking the crucible. The phosphorus is then precipitated in the usual way, and is determined and calculated to phosphorus pentoxide. This weight is deducted from the weight of the combined oxides already found, and the tin calculated.

This method will require at least two and one-half hours to get the phosphorus precipitated. It is necessary to run a blank on the potassium cyanide unless a special phosphorus-free cyanide is used. The amount of tin must always be sufficient to hold up all the phosphorus; that is, 6 per cent to 8 per cent of tin for 1 per cent of phosphorus, otherwise it is necessary to add sufficient tin or some of the phosphorus will go in the filtrate with the copper, lead, etc.

Another method quite frequently used is to make a sodium carbonate and sulphur fusion of the combined oxides of tin and phosphorus, leach out with water, and filter off any sulphide of copper which may have been held with the tin. Make the filtrate slightly acid with hydrochloric acid, and filter off the sulphide of tin, boil off excess hydrogen sulphide, add nitric acid, evaporate to a low bulk, cool, add excess ammonia, then nitric acid, and precipitate the phosphorus.

This method is by no means shorter than the previous one, and it is next to impossible to wash the sulphide precipitates free from phosphorus.

A modification of the previous method is used by some chemists. An aquia regia solution of the alloy is made, excess ammonia added, and the solution saturated with hydrogen sulphide. The sulphides of copper and lead are filtered off, then the filtrate is made acid with hydrochloric acid, and the tin sulphide removed. The phosphorus is then determined in this filtrate.

This method is the least satisfactory of any. It is next to impossible to wash the phosphorus out of the sulphide precipitates.

One can readily see that the foregoing methods are long and rather objectionable. With the idea of saving time and at the same time increasing the accuracy the writer conceived the idea of using cerium chloride as a precipitant for the phosphorus, similar to the one suggested by him for the determination of phosphorus in vanadium steels. (This journal, January, 1913.)

The method used is as follows: Dissolve one gram of the bronze in 10 c.c. conc. hydrochloric acid and 5 c.c. conc. nitric acid in a tall-form 300 c.c. beaker. When solution is complete, add 150 c.c. of hot water, and 10 c.c. of a solution of cerium chloride (1 gram cerium chloride, 25 c.c. hydrochloric acid, 200 c.c. water). Now add a solution of one part ammonium hydroxide and two parts water, from a burette, stirring continually until the solution has a greenish blue cast. This will require from 25 c.c. to 35 c.c. Now add 4 c.c. of acetic acid, and boil for five or ten minutes; remove from the hot plate and allow the precipitate to settle; this will require several minutes. Syphon off as much of the clear solution as possible, add 100 c.c. of hot water and again syphon. Repeat this washing six or eight times; with careful manipulation all but about 30 c.c. of the liquid can be decanted each time without disturbing the precipitate. After sufficient washing add 15 c.c. of nitric acid and 3 c.c. of hydrochloric acid

TABLE I

Phosphorus added.	Phosphorus found.
.10%	.084
.10%	.089
.10%	.105
.20%	.208
.20%	.192
.20%	.189
.30%	.306
.30%	.297
.50%	.509
.50%	.492
.50%	.496
.50%	.490
.70%	.711
.70%	.693
.80%	.806
.80%	.792
1.00%	.997
1.00%	.984
1.00%	1.008
1.00%	.990

ACTUAL PHOSPHOR-BRONZES

Average of five analyses.	Cerium Chloride.
.068%	.061
	.065
	.069
	.070
.78%	.765
	.790
	.788
	.774
.79%	.796
	.793
	.791
.65%	.664
	.672
	.659
.813%	.822
	.827
	.818

Babcock-Hagmaier Laboratories,
Lackawanna, N. Y.

and heat until the precipitate has all dissolved. When solution is complete, cool, add 5 c.c. (1-1) ammonia, and precipitate the phosphorus.

By this method the writer has been able to have the phosphorus precipitated in duplicate on three samples in an hour. When this method is used the tin is determined on a separate sample either electrolytic, by titration, or from the combined oxides.

The following results were obtained on actual phosphorus-bronzes, and on bronzes to which a known amount of phosphorus was added.

The Emissivity of Metals and Oxides

I.—Nickel Oxide (NiO) in the Range 609 to 1300° C.*

BY G. K. BURGESS AND P. D. FOOTE

The object of the present investigation has been the determination of the monochromatic emissivity E_λ and total emissivity E of nickel oxide NiO in the range 600 deg. C. to 1300 deg. C. This oxide forms a tough, smooth layer on the surface of nickel when subjected to high temperatures in air. Two methods were employed for the determination of the emissivity for red

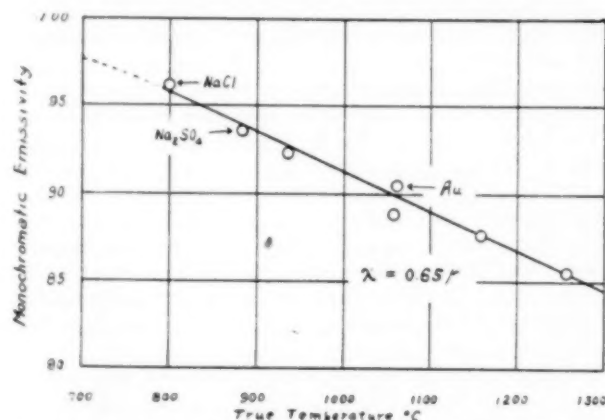


FIG. 1—MONOCHROMATIC EMISSIVITY ($\lambda = 0.65 \mu$) VS. TEMPERATURE

light: (a) that of microscopic melts and (b) a direct comparison by a spectrophotometer of the intensity of light emitted by the glowing NiO and by a black body at the same temperature.

In the method of microscopic melts, minute specimens of NaCl, Na₂SO₄, and Au were placed on the oxide which was electrically heated until these substances melted, as observed with a microscope, and the "apparent" temperatures at the instant of melting were measured by an optical pyrometer. The relation between the true temperature of the oxide, the observed or apparent temperature, and the emissivity is expressed by:

$$\frac{1}{T} - \frac{1}{S_\lambda} = \frac{\lambda}{0.4343 C_2} \log A_\lambda$$

where T is the true absolute temperature, S_λ the observed absolute black body temperature for a wave length λ , c_2

the Planck or Wien constant (14,450) and A_λ the emissivity for a wave length λ .

In the spectrophotometric method, a nickel strip was folded, forming a wedge or V-shaped cavity, and electrically heated until a coat of the oxide was formed. It has been shown by Mendenhall that the radiation from the interior of a V-shaped cavity is closely "black." The ratio of the intensities of the outside and inside of the wedge, properly corrected for temperature gradient through the walls, gives at once the emissivity at any

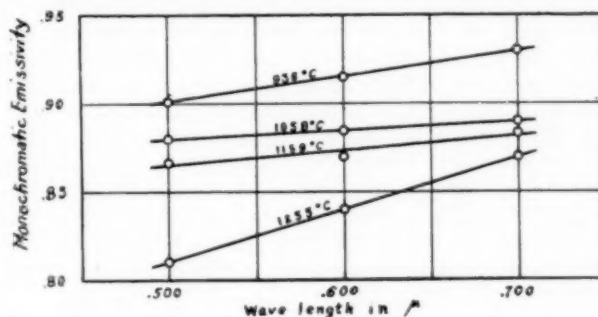


FIG. 2—MONOCHROMATIC EMISSIVITY VS. WAVE LENGTH

particular wave length and temperature. Determinations by this method agree with those by the method of microscopic melts, as is seen in Fig. 1. This curve represents the monochromatic emissivity, for $\lambda = 0.65 \mu$, as a function of the temperature. Over the range 700 deg. to 1300 deg. C., the emissivity decreases with increasing temperature.

Fig. 2 shows the dispersion of the emissivity of NiO in the range $\lambda = 0.5$ to 0.7μ for the temperatures 938, 1058, 1159 and 1255 deg. C. Over this range of the spectrum the emissivity is found to increase with increasing wave length. The type of curves suggests the presence of an absorption band in the early infra-red, but one which is probably broad and not well defined.

The total radiation of NiO was investigated by means of twelve radiation pyrometers of the Thwing and Féry types. These instruments were sighted upon a strip of the electrically heated oxide. The apparent temperatures of the oxide for $\lambda = 0.65 \mu$ were measured by a Holborn-Kurlbaum pyrometer and these values corrected to true temperatures by means of the determina-

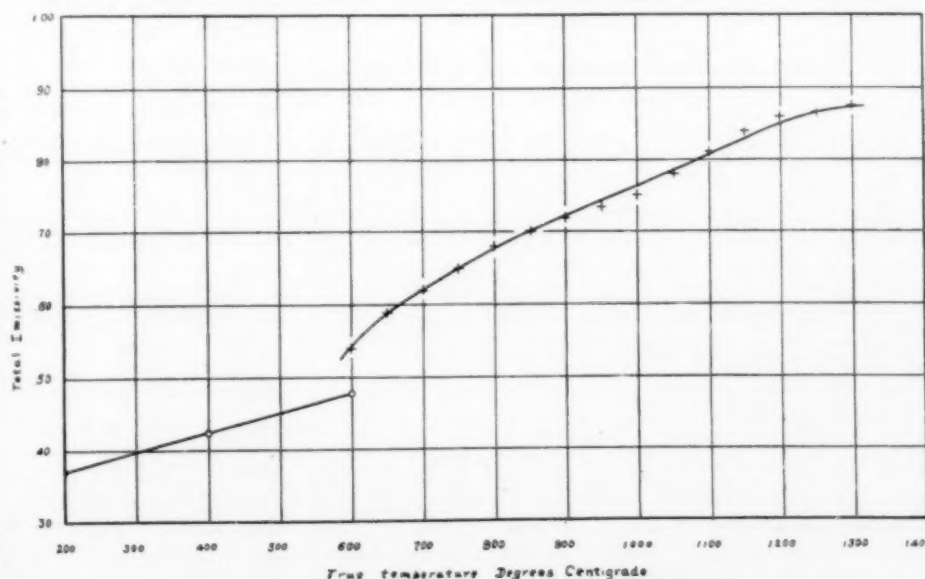


FIG. 3—TOTAL EMISSIVITY VS. TEMPERATURE

*The complete paper giving all observations and detailed description of apparatus will appear in the Bulletin of the Bureau of Standards.

tions on the monochromatic emissivity. Knowing the apparent absolute temperature, S , determined by the

radiation pyrometers, and the true absolute temperature, T , the total emissivity, E , may be computed from the relation $E = \frac{S - T_0^4}{T^4 - T_0^4}$, where T_0 is the absolute temperature of the receiver.

Fig. 3 represents the total emissivity as a function of the temperature. The increase of emissivity with increasing temperature is very pronounced. In the range 700 deg. to 1200 deg. the increase is nearly linear with temperature. Obviously, this linear relation will not extend indefinitely, for the emissivity can not exceed 1.00 or become zero. The characteristic curve must therefore be bent at both ends. This departure from the straight line is apparent at 1200 deg. to 1300 deg. It is possible that the emissivity at still higher temperatures increases very slightly and gradually until the melting of the oxide may bring about a discontinuity in the curve. Somewhere below 600 deg. the curve must again change its slope, possibly becoming approximately parallel to the axis of temperatures.

The three points represented in Fig. 3 by circles are from the data of Randolph and Overholser. It is possible to extend our curve so that it will coincide with the 400 deg. and 200 deg. points determined by these investigators. The disagreement of the two curves at 600 deg., however, would indicate a systematic difference between the two sets of data.

The following table gives the true temperatures corresponding to the temperatures observed by (1) an optical pyrometer using red light ($\lambda = 0.65 \mu$) and (2) by a total radiation pyrometer when sighted on nickel oxide.

Observed temperature deg. C.	Optical pyrometer true temperature deg. C.	Radiation pyrometer true temperature deg. C.
600	...	710
700	701	800
800	802	895
900	904	985
1000	1007	1075
1100	1110	1165
1200	1215	1255
1300	1320	1350

Bureau of Standards, Washington, D. C.

Energy of Vaporization of Carbon

BY CARL HERING

The energy required to vaporize carbon, commonly called the latent "heat" of vaporization, does not seem to have been determined experimentally. It is sometimes very desirable to know it in analyzing the energy reactions in electric furnace processes, in which the cost of the energy is often an item of importance. One therefore has to resort to the less satisfactory method of determining it by calculation, at least approximately. The following is an attempt to do so on a basis which may possibly be new:

According to experimental determinations the energy set free when one gram molecule (28 grams) of CO is formed from one gram atom (12 grams) of solid C and one gram atom (16 grams) of gaseous O, is given in books as 26.1 kilogram calories. When one gram molecule (44 grams) of CO₂ is formed from solid C and gaseous O, the energy set free is stated to be 94.3 kg cal. When one gram molecule of gaseous CO is burned to gaseous CO₂, 68.2 kg cal are set free, this being of course equal to the difference between the other two, 94.3 — 26.1.

When solid carbon is changed into a gas, then the attraction of the molecules of the carbon which caused it

to be a solid, has been overcome and the molecules have been separated far enough to make it a gas; this is true whether this gas is a true vapor of carbon or a gaseous compound of carbon. It takes energy to do this, and the amount required for the true vapor is usually termed the heat of vaporization. There seems to be no good reason why this should be materially different in amount when the gas is a compound of carbon like CO or CO₂, than when it is the pure vapor; the disruption of the molecules from each other is accomplished in either case. Let this energy be represented by V in kg cal for the amount of carbon now involved, namely, 12 grams.

The energy for this physical change of state must have been supplied by the energy of the chemical union of solid C and gaseous O, hence the observed 26.1 effective kg cal of their union to CO is only the difference between the purely chemical energy and what might be called the purely physical energy. The real chemical energy is therefore $26.1 + V$. Similarly the real chemical energy of $C + O_2 = CO_2$ is $94.3 + V$. When CO is burned to CO₂, no energy of vaporization is involved, all the components being gases before and after the reaction. The energy involved in changes of volume against atmospheric pressure, may be neglected in the present discussion as being relatively quite small; moreover, it is often considered to be included in the figures for the latent heats.

These two numerical values together with $CO + O = CO_2 = 68.21$, are not sufficient to calculate V from them. Prof. J. W. Richards in the *Trans. Amer. Electrochem. Soc.*, Vol. XIII, p. 451, estimates the latent heat of vaporization of carbon by calculation based on Trouton's statements, to be approximately 3808 calories per kilogram. But, like many other authors, he unfortunately omits stating whether these are large or small calories; there is no doubt, however, that kilogram calories were meant.

For the 12 grams of carbon now involved, the volatilization energy V , required in the combustion, would then be 45.7 kg cal. Hence the real chemical energy of $C + O = CO$ would be $26.1 + 45.7 = 71.8$; and of $C + O_2 = CO_2$, $94.3 + 45.7 = 140.0$. It will be noticed that when thus reduced to true chemical energies, the latter is almost exactly double the former, and the former is very nearly the same as the 68.2 for $CO + O = CO_2$. Changing the volatilization energy from 45.7 to 42.1 (only about 9 per cent) makes this relation exact; the true chemical energy of CO would then be 68.2, which is the same as found for $CO + O$; and that of CO₂ would be 136.4, which is exactly double 68.2.

While such an agreement may, of course, be merely a coincidence of no significance, yet on the other hand there is a good reason to believe that it should be so, as it would simply mean that, when reduced to true chemical energy, carbon develops exactly as much energy in combining with its first molecule of oxygen as it does in combining with the second one, and this would by no means be unlikely or irrational, but is even likely to be the case.

If the latter is true, then it becomes possible to deduce the true energy of vaporization of solid carbon very simply and directly from these two observed physical constants 26.1 and 94.3, in the way explained above. That for 12 grams then being 42.1 kg cal, the specific value would be 3510 kg cal per kg. This is only about 9 per cent less than Prof. Richards' calculated value 3808, which he himself gives only as an approximation. Bertholet, in his well-known treatise, does not give any definite value for the vaporization energy of carbon, but he states with positiveness that it is not less than 42.1 kg cal for 12 gr.; this is exactly the same figure as that given above.

Working backwards from this value by the same

method as used by Prof. Richards to deduce it, brings the boiling point of carbon (namely, the temperature of the arc) down to 3387° C., which seems to agree even better with other determinations than the value assumed by him, 3700° C.

To further investigate whether it is a mere coincidence or whether it is likely to be a possible law, that the true chemical energy of each bond between like elements is the same, the writer has in the accompanying table deduced the true chemical energy between the bonds of carbon and hydrogen, as the known compounds formed between these two elements are far more numerous than between carbon and oxygen. Those here given are only those which normally are gases, so as to eliminate the unknown physical energies which must be set free when gaseous hydrogen contracts in forming liquid or solid compounds. They are furthermore limited to those whose energies of combinations have been determined by experiment, or possibly only by calculation, and have been published. In the formation of all of these compounds it is here assumed (although not always so stated in the books) that the original materials were solid carbon and gaseous hydrogen, hence that a physical change from solid carbon to a gaseous compound is involved in the reaction. The so-called "heats" of combination on which the present deductions are based, are probably not accurate and may in some cases even be quite inaccurate, more especially in the more complex quantities toward the bottom of the table; hence very close agreement cannot be expected in such an analysis as this.

These fourteen different compounds of C and H are arranged in this table in the order of the number of their atoms of carbon and hydrogen. The first column gives their molecular formula (which in some of the more complex cases may perhaps not be definitely known). The second column gives the energies of their formation from solid carbon and gaseous hydrogen, as given in reference books; the numbers in all the other columns except the last, have been deduced by the writer. The + and - signs mean energies set free or consumed, respectively.

The third column gives the physical energies consumed in the vaporization of the carbon, using the basic figure deduced above, namely, 42.1 kg cal per gram molecule (12 grams) of carbon. This is the only physical change considered for the first approximation. The important point at present is not so much the actual value of this constant, as the fact that it is taken to be the same for all.

The next column shows what the purely chemical energy must have been so as to have furnished the physical energy in the previous column, and the net balance given in the second column; these net balances are the algebraic sums of the figures in the third and fourth columns.

Having thus determined the first approximation to the true chemical energy, these numbers are then divided by the number of hydrogen atoms to see whether the energy value for each hydrogen atom is the same in the different compounds. These results are given in the fifth column, and it will be noticed that, at least for the simpler and better known compounds, there is a rather surprisingly close agreement, considering the crudity of the basic data and the following additional correction factors.

It will be noticed that the energy per H atom thus deduced grows larger as the number of atoms of carbon in the molecule increases; also, in each group, as the number of hydrogen atoms per molecule increases; hence it would be natural to suppose that it might increase as their product. This increase is as would be

TABLE

I	II	III	IV	V	VI	VII	VIII
KILOGRAM CALORIES							
	PER GRAM MOLECULE			Chemical per H. Atom	Molecular per Gr. Mol. <i>e</i>	Second Approx. Col. V.	Combustion Energy per Gr. Mol.
	Net	Physical	Chemical				
CH_4	+18.9	- 42.1	+ 61.0	+15.25	0	+14.25	213.5
C_2H_2	-51.4	- 84.2	+ 32.8	+16.4	+ 2.3	+14.4	313.8
C_2H_4	-14.6	- 84.2	+ 69.6	+17.4	+ 8.6	+15.4	341.1
C_2H_6	+23.3	- 84.2	+107.5	+17.9	+16.0	+15.9	372.3
C_3H_4	-52.6	-126.3	+ 73.7	+18.4	+12.7	+15.4	473.0
C_3H_6	-17.1	-126.3	+109.2	+18.2	+17.7	+15.2	507.0
C_3H_8	- 9.4	-126.3	+116.9	+19.5	+25.4	+16.4	499.3
C_4H_8	+30.5	-126.3	+156.8	+19.6	+34.8	+16.6	528.4
C_4H_{10}	+ 2.6	-168.4	+171.0	+21.4	+49.0	+17.3	650.2
C_4H_{12}	+35.0	-168.4	+203.4	+20.3	+50.9	+16.3	687.2
C_5H_{12}	+ 7.3	-210.5	+217.8	+21.8	+65.3	+16.8	804.0
C_6H_6	-11.3	-252.6	+241.3	+40.2	+149.8	+34.2	784.1
C_7H_8	- 5.4	-294.7	+289.3	+36.2	+127.3	+29.3	933.8
C_8H_{10}	- 6.8	-336.8	+330.0	+33.0	+177.5	+25.0	1084.3
$C_{10}H_{16}$	+12.2	-421.0	+433.2	+27.1	+189.2	+17.1	1473.3

expected. More atoms are crowded into a molecule in the more complex materials. Such more crowded molecules mean physically that either there has been less expansion of the solid, or some contraction of the gas, or both; either means a gain of energy (physical). These gains mean that the physical energies in the third column are numerically less, and that the calculated, purely chemical energies are therefore less than those given in the first approximation in the fourth column.

For the present purpose of analyzing the energy reactions only, it may be assumed that all the atoms of carbon were first separated by the energy represented by the latent heat of vaporization (third column) and then afterwards compressed or condensed again to the extent represented by the molecular formula in the first column; also that the hydrogen atoms, originally fully separated, have also been condensed to the indicated extent. Such condensations or contractions are exothermic; that is, they set free energy, and this energy, it is not unreasonable to suppose, should be greater the greater this condensation (which it will be seen from the table is the case), but according to what quantitative law is perhaps not yet known.

The sixth column shows what this physical energy of molecular condensation *e* would have to be in kg cal per gram molecule of the compound if the law above suggested is true, namely, that the true chemical energy (when separated from all the energy of physical changes) is a constant for each hydrogen atom with which the carbon has combined. If these energies could be calculated or measured, and were found to agree approximately with those in this column, it would establish this proposition and at the same time confirm the value for the energy of vaporization of carbon assumed in this analysis. The first value in the fifth column (+15.25) has in this first approximation been taken as a basis for calculating *e* because that compound contains the least carbon per molecule.

It will be noticed that the values of this extra energy *e* increase approximately in proportion to the product of the number of atoms of carbon and of hydrogen. This may again be a mere coincidence, but both the direction and the relative amounts are what one might expect, and that some energy would be set free by increasing the number of atoms in a molecule will probably not be questioned; it is claimed, for instance, that

carbon vapor has double the latent heat of vaporization when monatomic than when diatomic.

Adjusting the purely chemical energies in accordance with this proposition concerning the energy e , gives as a second approximation for the hydrogen bond the figures in the seventh column, which it will be seen are in rather strikingly close agreement, excepting only some of the more complex molecules the constitution of which may perhaps not yet be definitely known. The agreement is now far better than in the fifth column.

In a further approximation the values of e should be added algebraically to the physical energy in the third column, in order to give the total physical energy involved; and they should also be subtracted arithmetically from the chemical energy in the fourth column, to give the true chemical energy; the algebraic sum of these new physical and chemical energies will then again give the observed net values in the second column.

It may or may not be a further coincidence that the purely chemical bond of oxygen is nearly the same for the combination with carbon as with hydrogen.

As a matter of interest the energies of combustion (presumably into CO_2 and H_2O , although not always so stated in books) are added in the final column. It will be noticed, and this may also be of significance, that when these compounds are arranged in this order, as distinguished from the order more generally used in books, they are also in the order of their combustion energy value, the only exception being again C_6H_6 (benzol) which is out of agreement in other columns also, hence is likely to be wrong; and either one of the two values of C_3H_8 (propylene and trimethylene), which, disagreeing with each other, cannot both be correct.

The above analysis is an illustration of the possible importance of more clearly distinguishing between the purely physical and the purely chemical reactions in energy equations, as was advocated in an article on this subject by the writer in the July issue of this journal, page 439.

The writer is indebted to Prof. G. A. Roush for some information concerning existing and non-existing data.

Philadelphia, Pa.

Recent Chemical and Metallurgical Patents

Iron and Steel

Electric Steel.—To Mr. William R. Walker, of the United States Steel Corporation, recently two interesting patents have been granted on electric steel processes.

The first patent relates to the manufacture of steel from iron which may be classified as to its phosphorus content between the grades commonly known as basic Bessemer iron (1.8 to 2.2% P) and acid Bessemer iron (below 0.12% P). The iron is first blown in acid converters of 13 tons capacity to remove the silicon and a portion of the carbon. The iron is only slightly oxidized by this partial blow. It is then transferred to an intermediate mixer of 125 tons capacity in form of a basic open-hearth furnace, so as to bring about substantial uniformity in the products of the several converters and to carry always a supply of hot metal for the subsequent treatment in the electric furnaces. In the mixer the phosphorus content is reduced to the point desired in the final product, while a portion of sulphur is incidentally removed and more iron is oxidized. From time to time portions of metal in the intermediate mixer are transferred to acid-lined electric furnaces of 25 tons capacity where the metal is deoxidized. The metal is then transferred to a final mixer in form of a ladle of 100 tons capacity with an acid lining and with or without electrodes or other heating

means. A substantial quantity of molten metal is maintained in this final mixer so as to obtain a product uniform in carbon and other ingredients. The refined steel produced in this final mixer is then cast either directly therefrom or by transferring it first to a more convenient smaller casting ladle. Either in the electric furnace or in the final mixer or in both, deoxidizing additions of manganese and aluminium are made to the metal. The chief feature of this combination process is that the cost of the electric furnace treatment is considerably reduced. Not only does the intermediate mixer provide metal for a nearly continuous use of the electric furnaces but the cost of dephosphorization in the mixer is low. The electric furnace is used for nothing but deoxidation for which purpose it is eminently fitted on account of its reducing atmosphere. Finally the use of an acid electric furnace means a substantial saving of expense, both first cost and repairs and other operating costs, and a larger tonnage per day than could be obtained with a basic lining. (1,086,003, Feb. 3, 1914.)

In Mr. Walker's second patent, which refers to basic-lined electric furnaces, it is pointed out that the object for using a highly basic slag in electric steel furnaces is to reduce to a minimum the sulphur content of the product. He has found, however, that by using a slag containing a comparatively high percentage of silica, the sulphur content may be reduced to practically the same extent as before and considerable economies are effected in the operation of the furnace and also an improvement obtained in the quality of the steel. Mr. Walker proposes to increase the quantity of silica in the slag in basic processes to such an extent that a part will be reduced and remain in the metal, at the same time keeping the percentage of silica so low that it will not cut the hearth to any substantial extent. As a good practical example is cited a slag made of silica 30 per cent and lime (approximately pure CaO) 70 per cent, to which is added coke at least 5 per cent by weight of the combined silica and lime. This slag will also contain magnesia and alumina, which are derived from the furnace bottom materials and the lime additions. The process has various advantages. Slag of this composition is more fusible. The quantity of fluorspar may be reduced to little or none. The formation of fumes in the furnace is minimized. The silica roof of the furnace will last much longer. The process is of particular advantage in the manufacture of soft steel. The greater or less quantity of nascent silicon which enters the bath serves to purify it and in a way to make up in purifying effect for the lack of carbon in the bath. The silica in the slag may be so proportioned that the quantity of nascent silicon which enters the bath shall be only sufficient to have a purifying effect and shall not be sufficient to leave any appreciable percentage of silicon in the finished product. (1,086,489, Feb. 10, 1914.)

Manufacture of Electrolytic Iron.—To obtain high efficiency in the electrolytic production of iron deposits, Anthelme Boucher suggests the use of a high-current density. Ferrous salts may be used as electrolyte and to obtain a smooth surface, the electrolyte is stirred with air, whereby the ferrous chloride is changed into an oxychloride of iron. "This reacts on the hydrogen in bubbles adhering to the cathode, and thereby eliminates the cause of the bad quality of the deposits." The electrodes, therefore, are rotated. Besides the oxychloride, ferric chloride is also produced. To prevent the ferric chloride to be converted back into ferrous-salts the electrolyte is passed in a separate cell over iron shavings. The solution should become chestnut brown and not foaming. The higher the current density employed the greater must be the

speed of the rotating cathode and the higher must be the temperature. At 1000 amperes per square meter the suitable temperature is 75 to 77 deg. C. The concentration of the liquid must be constant and be chosen according to its depolarizing power; it is maintained by adding fresh liquid to it. The greater the speed of circulation of the electrolyte, the lower will be the proportion of phosphorus. Air can be injected in varying quantities, forming basic salts for the depolarization. (1,086,132, February 3, 1914.)

Manufacture of Iron.—Augustin E. Bourcoud, of London, England, wants to reduce iron from its ore by means of gases. The CO_2 resulting is regenerated again. Comminuted or granulated ore and flux travel under agitation through a stream of CO and some CO_2 , and the reduced metal is then melted in an electric furnace. Injurious moisture, organic and volatile impurities are eliminated by hot air, hydrogen is eliminated by coke in a regenerator, sulphuric gases by lime above 900 deg. C. Sulphur in metal is removed in the electric melting furnace. The furnace is arranged to utilize all waste heat; it controls temperature and composition of gases, waste products and charge. In Fig. 1 producer gas from o is burned in the stove

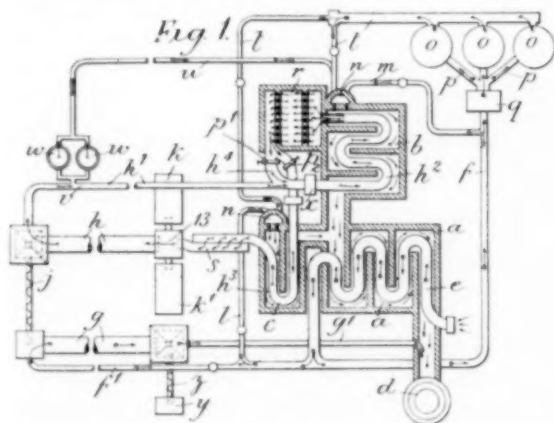


FIG. 1—IRON REDUCTION BY MEANS OF GASES

inlets n , the gases leaving through chimney d . Air is highly heated in stove a and distributed to supply the producers o , to burn gas for heating stoves, while a third part passes through the inclined rotary heater g to the chimney d . The charge enters g in opposite direction, where it is dried. The hot producer gases then reduce it in the reduction furnace h . The gases, now rich in CO , are highly heated in stove b and regenerated with coke in i , heated again in c , desulphurized in s and used over again in furnace h . The reduced iron is then discharged into either electric furnace k or k' by slide valves. (1,086,133 and 1,086,134, February 3, 1914.)

Refining Steel.—Low-carbon steel can be made or dephosphorized in an electric or open-hearth furnace by an oxidizing slag when adding manganese from time to time into the bath, according to a patent of Axel Hethey, of London, England. The charge should contain from 0.25 to 1.00 per cent Mn , which prevents a solution of iron oxide in the metal, yet oxidizes the phosphorus and removes also oxygen and sulphur. According to another feature of this process, the oxidizing slag may be left in the furnace and to prevent oxide being dissolved, the slag is stiffened with lime or dolomite, so that the heat can be finished with increased temperature to render pouring easier. (1,089,410, March 10, 1914.)

Furnace Roof.—A water-cooled flat roof, especially adapted for open-hearth furnaces was invented and

patented by Edwin E. Slick, of Pittsburgh, Pa. Ordinary buckstaves support a series of water-cooled boxes, forming supports or girders. The bottom of said boxes has a dove-tail groove, co-acting with fire-bricks of similar shaped projections. The upper diagram of Fig. 2 shows how the remaining space is

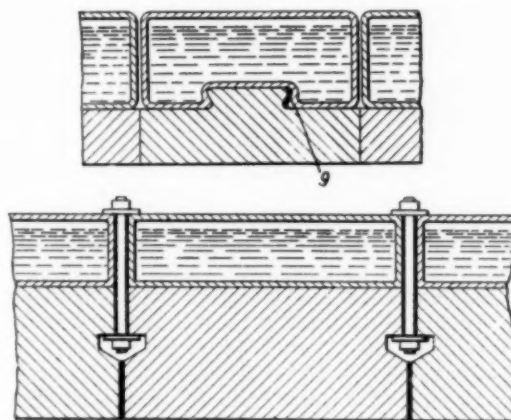


FIG. 2—FURNACE ROOF

filled with small bricks or mortar, allowing the fire-bricks being replaced independently. The lower diagram is a modified form, in which the recesses allow suspension by means of washers and bolts from the water-cooled boxes. (1,088,634, February 24, 1914.)

Reduction Without Smelting.—Among the proposals to reduce iron and other oxide ores without resorting to smelting, is that of Mr. Harvey C. Alford, of St. Louis, Mo. His patented process consists in heating externally a charge of ore in a finely divided state, to a temperature below the point of fusion, mixing the heated ore with powdered coal, without access of air, and then agitating and heating this mixture in a suitable container to a temperature approximating 1000° F. The operations are carried out in revolving drums, the discharge of the first mixing with coal and entering the second. An excess of carbonaceous fuel is maintained to avoid the occurrence of reversible reactions between the reduced metal and any source of oxygen such as carbon dioxide. The product is reduced metal or sponge, mixed with the excess of fuel which may be removed from the metal in any suitable manner. (1,097,156, May 19, 1914.)

Manufacture of Steel.—According to a process of refining steel, whereby phosphorus, oxides, silicates and other impurities are removed, patented by Mr. Albert Patton, of McKeesport, Pa., metal that has been blown in the Bessemer converter, in which carbon, silicon and manganese have been reduced as low as possible, is treated as follows: Pour into an acid-lined intermediate receptacle, and from this into a basic-lined receptacle, care being taken to prevent the acid slag from entering the basic-lined receptacle. Materials for forming a basic slag are added to the latter before or during the pour. Phosphorus, being present in the Bessemer metal as P_2O_5 , is absorbed by the basic slag. Manganese or re-carburizer can be added, if desired, in the basic-lined receptacle. (1,091,588, March 31, 1914.)

Bessemer Converter.—According to an invention of Mr. Joseph V. Bretaud, of Highland Park, Mich., the cost of construction and operation of a converter plant may be reduced by using converter shells comprising an upper and a lower section. Each section is removable from its support independently of the other, so that when the lining gives way in one part, it is neces-

sary only to remove that section in which the break occurs. The removed section, having its lining exposed, cools rapidly and can be repaired quickly. The inventor claims that with this construction, two converters can produce the same number of heats otherwise obtained from three ordinary converters. (1,088,401, Feb. 24, 1914.)

Gold and Silver

Apparatus and Method for Filtering Slimes.—A number of patents recently granted to Dr. Charles Butters, of Oakland, Cal., disclose his latest ideas in slime filtration, and particularly his methods of avoiding conflict with the ideas of other inventors regarding dislodgment of the filter cake from the fabric. Heretofore it has been the practice to effect dislodgment by air pressure applied internally to the leaf, and it has been assumed that such pressure was necessary to secure dislodgment in economical time. This application of compressed air has resulted in certain requirements in the construction of filter leaves, notably the use of outside strips fastened at intervals to the filtering medium, or stitching the two sides of the leaf together, for the purpose of preventing ballooning when internal pressure was applied. This manner of construction has resulted in reducing the available filtering area of the leaf, as well as weakening it and shortening its life.

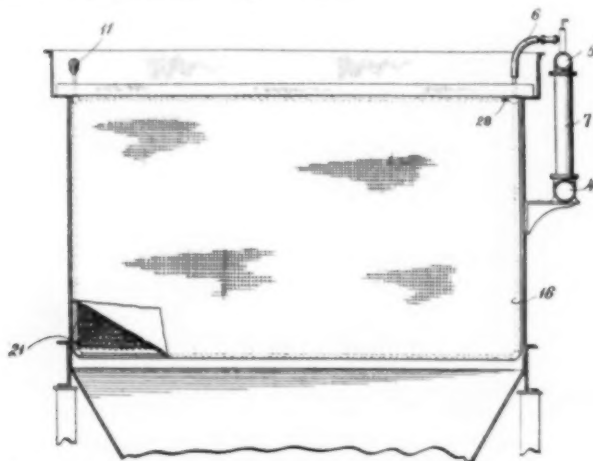


FIG. 3—NEW VACUUM FILTER LEAF

Mr. Butters has discovered that internal pressure is not necessary to the dislodgment of the cake, and that this operation can be accomplished by saturating the entire area of the filter medium with water, thereby destroying the adhesion of the cake. Water is applied internally, but not under pressure, and a relief valve is provided to permit the complete expulsion of air as the leaf fills with water. "It is understood that the cake may be, and preferably is, dislodged while the leaf is submerged in wash water. This means that there will be considerable hydrostatic pressure on the outside of the leaf. It has been found by experiment that the difference in specific gravity of the more or less dirty tank wash water and the relatively clean cake-dislodging water would be sufficient to maintain a column of the relatively clean water inside the leaf several inches higher than the level of the tank wash water, so that the relief valve (for expulsion of air) and the inlet for the cake-dislodging water may be several inches above the surface of the tank wash water without producing greater pressure inside the leaf than the outside pressure on the leaf."

By this means the external hydrostatic pressure may be balanced but can never be exceeded by the

internal hydrostatic pressure, and yet the fabric may be moistened by water applied internally, and the cake easily dislodged. This system of treatment also makes it possible to use a lighter filter fabric and to dispense with bracing, stitching, etc., formerly used to prevent ballooning.

The construction of the filter leaf is shown in the drawing Fig. 2, in which a frame of pipe or tubing is covered with a bag of duck or other fabric. Inside the bag is a filler of cocoa matting to keep the sides of the bag apart when vacuum is applied. The lower horizontal member of the frame is perforated or slotted along its upper side, and is connected to a vacuum pump by means of the pipe 16 and flexible connection 6. The upper horizontal member also is perforated or slotted along its lower side, and connected with the air-relief valve 11. Plugs are placed in the frame at 21 and 28. The air-relief valve is of any kind that will prevent the ingress of air when the leaf is under vacuum, but will allow air to escape when the leaf is filled with water.

In operation, a cake is formed on the leaf, and washed in the usual manner. When it is ready to be dislodged, water is admitted to the leaf by way of pipes 4, 7, 5 and 6, filling the leaf from the perforations or slot in the lower horizontal frame-member. As the water rises in the leaf the air is expelled through the relief valve so that the water finally fills the leaf completely and saturates the fabric. The cake is automatically and rapidly dislodged. (1,100,217-18-19-20-21-22-23-67, June 16, 1914.)

Cleaning Filter Leaves with Lactic Acid.—The use of a solution of lactic acid to remove deposits of lime from filter leaves has been patented by Mr. Paul F. Leach, of Wollaston, Mass. The patent is assigned to Moore Filter Co., of New York City. The inventor states that a solution of lactic acid in the proportion of 1 lb. of the commercial product, which is of 22 per cent strength, to 25 lb. of water, gives satisfactory results. Heating the solution to about 150 deg. F. accelerates the dissolution of the lime. Cold solutions will cleanse the leaves in about an hour and hot solutions in half that time. The fabric is not injured in any way; its fibres remain soft and flexible, and are not shrunk by the action of the lactic acid. (1,101,639, June 30, 1914.)

Apparatus for Precipitating Metals from Solution.—According to the specifications of a patent granted to Mr. Frank A. Ross, of Spokane, Wash., and assigned to Chalmers & Williams, of Chicago Heights, Ill., the inventor proposes to precipitate, say gold-bearing cyanide solution by causing it to flow successively through a number of revolving cylinders containing solid precipitant in the form of aluminium. The cylinders are perforated, and mounted on hollow, perforated shafts in wooden tanks. The solution flowing into the first tank of the series fills the cylinder where it is acted on by the precipitant. It flows out by way of the perforated hollow shaft and thence into the next box where the operation is repeated. Electrodes may be placed in each compartment, the idea being that electrolytic action will aid in precipitation. (1,101,569, June 30, 1914.)

Zinc

Controlling the Formation of Zinc Dust in Smelting.—Preheating the smelting charge before placing it in the retort, is the method proposed and patented by Alex Roitzheim, of Düren, Germany, for preventing or minimizing the production of zinc dust in the smelting of zinc ores. He states that zinc dust is formed mainly in the first three to six hours after charging the retorts, and mainly before the beginning of the

reduction of the charge. This he ascribes to the fact that when the mixture is placed in the retorts in a cold condition, the reduction commences at the outer portions where the charge is in contact with the hot walls of the retorts. The zinc vapor there produced passes through the cold interior of the charge on its way to the condenser, and becomes suspended as zinc dust, entering the condenser as such. When the entire charge becomes heated, the zinc fume begins to condense as liquid. He therefore proposes to preheat the charge to a temperature of 600 deg. C., which is between the temperature at which reduction commences and condensation occurs. (1,100,490, June 16, 1914.)

Electric Zinc Smelting.—According to the specifications of a patent granted to Mr. Heinrich Specketer, of Griesheim-on-the-Main, Germany, one of the sources of difficulty in the electric smelting of zinc where the charge is used as a resistor, lies in the fact that the bottom of the charge becomes a better conductor than the overlying layers, due to the greater density of the former portion. This results in an uneven reduction of the mass. In order to obviate this difficulty Mr. Specketer proposes to use a rotatable electric furnace in which the position of the charge can be shifted from time to time, thereby maintaining more even conditions of resistance and smelting. Electrodes are placed at each end of the furnace, one near the exit for fumes and gases and the other at the opposite end. Both electrodes extend over the whole inner end surfaces of the furnace and have their connections disposed 180° apart around the circumference of the furnace. (1,099,211, June 9, 1914.)

Copper, Lead

Separation of Copper from Nickel and Cobalt.—Mixtures of minerals of copper, nickel and cobalt have presented difficulties in metallurgy that have not been wholly overcome. Mr. Victor Hybinette, of Fredericktown, Mo., recognizing the problem presented by this mineral combination, has invented and patented a process which he claims affords a successful method of treatment. "This ore consists of a mixture of chalcopryrite and linnæite or siegenite, with silica and dolomite as gangue. There are also present varying quantities of marcasite and galena. An average sample of this ore will analyze, chalcopryrite 10 to 20 per cent; linnæite 4 to 8 per cent; galena 1 to 6 per cent; marcasite 2 to 10 per cent; silica sandstone 50 to 70 per cent, and dolomite 10 to 30 per cent." If lead is present in sufficient quantity, the inventor prefers to remove it by ordinary wet concentration, after which he treats the residue according to the following method, which would be adopted if lead were not present in large quantities. The ore is ground to 10-mesh or finer, roasted to remove about one-half the sulphur which is present in the chalcopryrite and innæite and subject to magnetic treatment. Copper and cobalt minerals are removed; the tailing will contain some copper and nickel and all the lead, if the last was not removed as above stated. These can then be recovered by wet concentration. The magnetic concentrate, containing 12 to 20 per cent copper, 2 to 5 per cent cobalt and 2 to 5 per cent nickel, is then smelted to matte, roasted to remove all or practically the sulphur and leached with sulphuric acid. Copper will be dissolved and nickel and cobalt will remain in the residue. Little or no iron will be dissolved from this roasted matte. (1,098,443, June 2, 1914.)

Filtration of Flue Dust from Furnace Gases.—For the purpose of removing condensable metallic fumes and other solid matter from the furnace gases of lead

and copper smelters, Mr. Gilbert Rigg, of Palmerton, Pa., proposes to use a device illustrated in Fig. 3. The patent is assigned to the New Jersey Zinc Co. The object of the apparatus is to provide a filter and condensing medium for the retention of solid matter carried in the furnace gas and to maintain the filtering medium in a condition for continuously performing its function. Referring to the drawing, A indicates a housing suitably supported and divided into compartments C, D and E by means of gratings consisting of a series of bars F of inverted V-shape and a series of flat bars G. Transverse rods I extend through the housing and are connected with rods K directly above the bars G. Mechanism for moving these rods I and connected rods K is shown at the left of the housing.

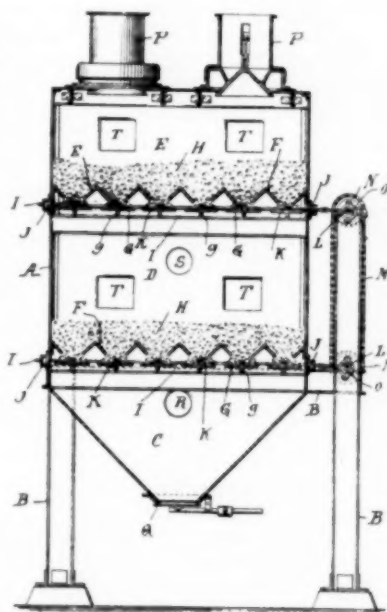


FIG. 4—FILTRATION OF BLAST FURNACE GASES

The housing is further provided with hoppers P at the top and an outlet Q at the bottom. The gas to be filtered enters through the inlet R into compartments C, and the filtered gas takes its exit from compartment E through the outlet S. The material resting on the gratings consists of coarse coal, coke, gravel, slag, stone, brick or the like. Accretions from the solid matter in the smelter fume build on the coarse material on the lower grating and form an efficient filter. When the accretions are so thick as to retard the draft, the mechanism is brought into operation, moving the rods K over the bars G, causing the accumulation to fall into the compartment below. At the same time an equal amount of fresh material is dropped from the upper grating. The operation of the mechanism may be arranged to be controlled automatically by a pressure regulator, or it may be left to the attendant who will regulate it according to the condition of the filter as indicated by the pressure. In some instances the reciprocation of the rods K may be made continuous. (1,095,676, May 5, 1914.)

Sintering Fine Ores

Use of Addition Agent in Sintering.—In the sintering process as devised by Mr. Ferdinand Heberlein, the mass of ore is exposed to the action of a current of air passed upwardly through the charge. A small percentage of the mass is not sintered, and in the case of agglomerating very fine dust it may happen that a considerable quantity may remain unsintered. To overcome this tendency Mr. Edmond A. Pretceille, of Nantes, France, proposes the addition of a salt, such as ferrous sulphate, to the charge, wetting the latter with a strong solution of the salt. It is his idea that the moisture thus added will be retained in the mass longer than if water only is added and that the salt will exercise a cementing effect on the fine particles, holding them together until they can be sintered by the heat of combustion. In the case of

iron ore it would suffice to add sulphuric acid, which would form iron sulphate with part of the ore. In the case of iron sulphate solutions, 10 to 15° Be. has been found a suitable concentration. (1,096,054, May 12, 1914.)

Improvement to Dwight-Lloyd Sintering Machine.—Mr. Eugene H. Laws, superintendent of the Ohio & Colorado Smelting & Refining Co. at Salida, Col., has patented an improved construction of the sintering machine made by the Dwight & Lloyd Sintering Co., New York. In machines of this type a continuous traveling grate of pallets passes over a suction box for the purpose of inducing a down draft through the charge contained in the pallets. The edges of the pallets rest directly on the edges of the suction box and the transverse portions of the pallets drag over dead plates at the ends of the suction box, in order to prevent air entering the box otherwise than through the charge on the pallets. It so happens, however, that the wear on the edges of the pallets is much greater than on the transverse portions, with the result that eventually the transverse portions occasionally engage and tear the dead plates and strain the driving mechanism. Mr. Laws provides an adjustable support for the pallets as they pass over the suction box, whereby each pallet is carried in close proximity with the edge of the box, but out of wearing engagement with the edges or dead plates. (1,097,592, May 19, 1914.)

Roasting Furnaces

An invention relating to the air-cooling of the central rotatable shaft of furnaces of the superimposed-hearth type, such as the Herreshoff, is patented by Mr. Edward J. Fowler, of San Francisco, Cal. The patent is assigned to the General Chemical Co., of New York City. Fig. 5 shows a cross-section of the shaft. It consists of a number of superposed sections fastened together by means of bolts. Concentric with the outer cylindrical wall, 3, is an inner cylindrical wall, 4, the two being held separate by radial ribs, 5. This structure forms an axial passage, 6, separated from and surrounded by an annular passage 7. Rabble arms used in connection with this shaft are hollow, and are divided into two interior passages by means of a longitudinal rib. When the arms are attached to the central shaft, one of the passages is in conjunction with the axial area, 6, and the other with the annular area, 7. Hence air passing up through the inner area of the central shaft is deflected through the opening 10, passes through the rabble arm and returns to the shaft by opening 9, and thence to the atmosphere. (1,100,426, June 16, 1914.)

An air-cooled rabble arm for use in roasting furnaces is patented by Mr. Utley Wedge, of Ardmore, Pa. It is shown partly in plan and horizontal section

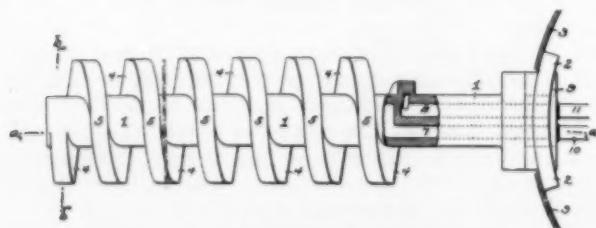


FIG. 6—AIR-COOLED RABBLE ARM

in Fig. 6. The rabbles or blades, 4, are cast integral with the arm. On the back of the arm and integral

with it are helical ribs, 5. Both the blades and ribs are hollow and connected interiorly, so that there is a continuous passage from the blade at the outer end of the arm to the one at the inner end. The rabble arm, 1, also is hollow, and communicates at its outer end with the hollow passage in the blades. By this construction it is possible for a current of air to flow from a passage in the central shaft to which the arm is attached, through the arm and back through the blades and thence to the air. (1,100,461, June 16, 1914.)

Electric Furnace Construction

Calcium Cyanamid.—Mr. Walter S. Landis, of the American Cyanamid Co., of Niagara Falls, N. Y., produces calcium cyanamid in apparatus shown in vertical and horizontal cross-section in Fig. 7. 1 is a furnace lined with refractory material and provided with

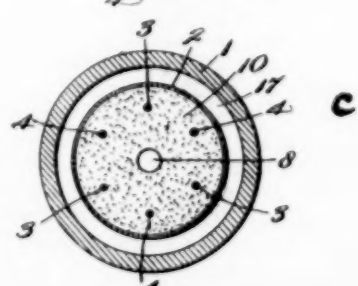
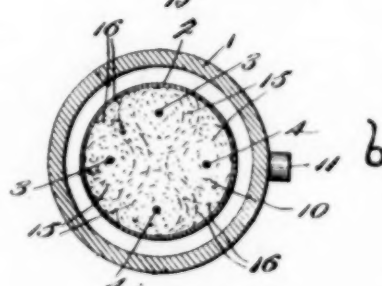
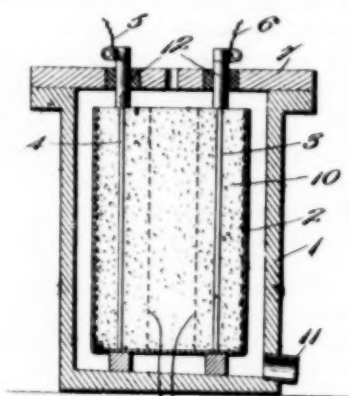


FIG. 7—CALCIUM CYANAMIDE FURNACE

a lid 7. Inside the furnace there is a foraminous vessel 2 with holes of $\frac{3}{4}$ or 1 inch diameter covered inside with a textile fabric to prevent the carbide from falling out. The nitrogen gas is admitted through 11 into the annular space between the outer furnace wall and the inner vessel and enters into the inner vessel through the open top and through the holes in the sides and bottom. In construction there is also a central hole provided for the passage of the nitrogen gas. 3 and 4 are a number of carbon resistors heated electrically to produce the desired reaction. Their connections 12 to the leads 5 and 6 are made of large diameter to heat them to a lower temperature and prevent their destruction by impurities in the nitrogen gas. 15 and 16 indicate the proceeding zones of reaction. With this construction the life of the resistors is very much prolonged. Further, with several resistors the reaction is much quicker than with a single one. (1,103,060, 1,103,061, 1,103,062, July 14, 1914.)

Electric Furnace.—The Rennerfelt furnace was described in some detail in our April issue, page 275. A patent has now been granted to Mr. Ivar Rennerfelt for the principal feature of the furnace, namely, the arrangement of the electrodes. The second claim refers to "the method of generating heat by means of electric arcs consisting in passing two-phase currents along and between two opposing electrodes to the end

of a third electrode at right angle to and symmetrically located between the first-named electrodes so as to cause an electric arc to burn between the ends of all the electrodes, strongly deviating away from the ends of the electrodes in the direction indicated by the symmetrically located electrode." (1,103,379, July 14, 1914.)

Abrasive Compounds.—A patent has been granted to **Thomas B. Allen**, of the Carborundum Company of Niagara Falls, N. Y., for the production of artificial garnet, an abrasive, by melting gradually 2240 lb. of calcined magnesite, 1493 lb. of alumina and 2760 lb. of silica. The molten mass is cooled slowly by insulation with some charge and when solid is removed at a temperature of 1200 to 1500 deg. C. and crushed while cooling with water to prevent dissociation. Similar compounds can be produced of the form $3R'OR''O_3SiO_2$, where R' is calcium, magnesium or ferrous iron and R'' is aluminium, ferric iron or chromium. (1,087,705, February 17, 1914.)

Electric Furnace.—**Bayard Guthrie** and **Joseph P. Karch**, of Pittsburgh, Pa., have invented a combined electric arc and resistance furnace. The crucible 5 is supported and insulated by fire brick from the bottom plate. The surrounding bricks have a pair of chambers 11, separated by a refractory partition 12-12 to prevent arcing between the electrodes and to hold the crucible when tilting. The electrodes 13-13 fit snugly to the crucible bottom and flare at the top, the space between is filled with broken carbon 14 to break up the electric arcs between electrodes and crucible.

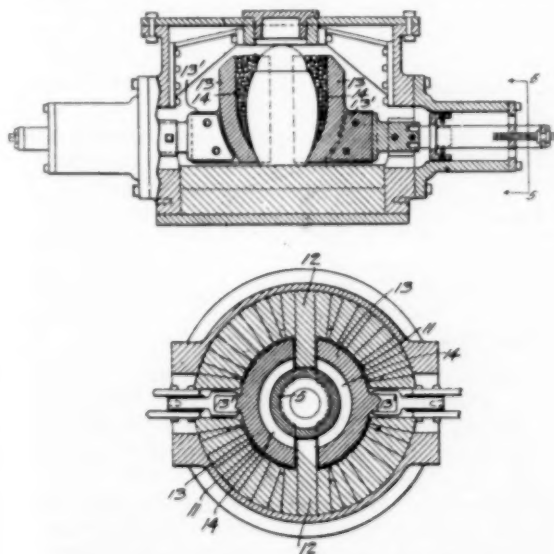


FIG. 8—ELECTRIC MELTING FURNACE

The electrode lugs 13' connect with copper electrodes and cables and an arrangement to adjust and hold the crucible by means of the electrodes. The heat is concentrated at the crucible top. This furnace is particularly adapted for direct or single-phase alternating current. The spout is so arranged as to discharge the metal direct into the center of a swinging mould, and touching its side walls. (1,086,164, February 3, 1914.)

Electrodes.—The fact that the energy losses through heat conduction in the electrodes of electric furnaces exceed those caused by electric resistance is made the object of a patent granted to **Dr. J. W. Richards**, Bethlehem, Pa. The electrodes are built with gaps extending inward in different shapes. These gaps are either left as open spaces or are filled with material of low heat conductivity and thus lengthen the path of heat currents to the outside of the electrode. For example,

it is possible to effect a large saving in electric energy by reducing the heat conductance 25 per cent., even though the electric conductance is thereby diminished 50 per cent. (1,088,296, February 24, 1914.)

Protective Coating for Electrodes.—The production of a coating of titanium nitride on furnace electrodes and other refractory materials is patented by **Mr. Samuel H. Fleming**, of Cleveland, O., and assigned to the National Carbon Co. of the same place. Titanium nitride is stable at high temperatures, is non-porous, tough and hard, and otherwise possesses qualities that make it valuable as a protective coating for substances that are subjected to very high temperatures in an electric or other furnace. In order to form the coating a mixture of rutile and some binder is pressed on the article to be coated, which is then baked at a high temperature in the presence of nitrogen. Owing to great affinity of titanium for nitrogen, the former is converted into nitride. A first coating of titanium carbide may be desired, as this substance wets carbon and enters the pores of its exterior surface and forms a perfect union with the carbon. This may be formed by applying a mix of titanium and carbon and heating with exclusion of air, forming titanium carbide. After the coating of carbide is formed the article may be heated in an atmosphere of nitrogen, thus converting part of the carbide into nitride. This latter conversion may be stopped before all of the carbide has been changed, if an inner coating of carbide is desired. (1,098,794, June 2, 1914.)

Aluminium Nitride Lining for Furnaces.—In order to overcome the great difficulties encountered in forming suitable linings for furnaces in which temperatures of, say 2000° C. are employed, **Mr. Ottokar Serpek**, of Paris, France, proposes the use of aluminium nitride. It possesses the property of being easily agglomerated and can be molded into various forms by using a binder such as sodium silicate. Such material resists the corrosive action of many chemical substances and may be advantageously used in industrial furnaces of different kinds. (1,099,131, June 2, 1914.)

Phosphoric Acid

A method of producing phosphoric acid from phosphate rock has been patented by **Mr. Frank S. Washburn**, president of the American Cyanamid Co., Nashville, Tenn. The process disclosed is an excellent illustration of the possibility of bringing about chemical reactions at temperatures obtainable only in the electric furnace, and of the economic value of a combination of fuel and electric furnaces in commercial operation.

The chemical basis of the operation is the displacement of phosphoric acid from its compounds by means of silica. It is found, however, that the replacement is far from complete at temperatures obtainable in fuel furnaces, say 1400 deg. C. On the other hand, if an electric furnace be used, it is possible at higher temperatures to greatly increase the yield of phosphoric acid, and if coke be added to the smelting charge, the evolution of phosphorus can be made almost quantitative. Nevertheless, the cost of electric energy is too great to permit of carrying out the process economically. Thus, phosphate rock contains from 10 per cent to 20 per cent of calcium carbonate. The carbon dioxide can be driven off from this compound by the heat evolved in burning from 0.012 to 0.025 ton of coal per ton of rock, whereas it would require from 200 to 300 hp-hr. of electrical energy for the same purpose. With coal \$3 per ton and electricity at \$15 per hp. year, the comparison of cost is as 7½ and 15 cents, respectively, are to 40 and 80 cents,

respectively. The heat in the slag, per ton of rock charged, is equivalent to 1600 hp-hr., or \$3.20 worth of electrical energy; while coke required for heating the charge is worth only \$2. Thus a combined process may be more economical than the electrical alone. Phosphate rock mixed with silica and a suitable fuel can be smelted in a small blast furnace, producing a slag which can then be further heated in an electric furnace, with coke added to complete the reduction of phosphorus. Mr. Washburn has further demonstrated that by the use of very rich oxygenated air, which is conveniently obtained as a by-product from a liquid air plant, he can get very fusible slags that can be tapped from the blast furnace without difficulty.

The gases evolved from both shaft and electric furnaces are led to a common oxidization chamber, and any unoxidized phosphorus is there condensed by absorption in water or ammonia. By the use of a combination of fuel and electric furnaces, it is possible to obtain an almost complete extraction of phosphorus from the rock, with an expenditure of electrical energy less than half that required by an electric furnace alone. Slags of 45 per cent silica offer no trouble in the shaft furnace when enriched air is used. (1,100,639, June 16, 1914.)

Electrolytic Processes

Sodium Hydroxide and Hydrochloric Acid from Brine by Electrolysis.—In a patent of the late Eugene A. Byrnes a diaphragm cell is proposed, divided up into a series of compartments containing alternately brine solution and water. In Fig. 9 the compartments 1, 3, 5 contain NaCl solution, the compartments 2 and 4 contain H_2O ; 1 contains the anode and 5 the cathode. The partitions between the successive compartments are used as bipolar electrodes. They may consist, for instance, of wire gauze, covered by porous diaphragms. At the partition between 1 and 2 the Na cations from 1 and the OH anions from 2 come to-

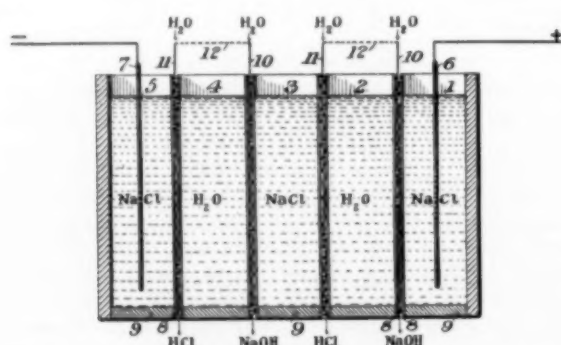


FIG. 9—SODIUM HYDROXIDE AND HYDROCHLORIC ACID FROM BRINE

gether and form Na OH which is drawn off. At the partition between 2 and 3 the H cations from 2 and the Cl anions from 3 come together and form HCl which is drawn off. The bipolar electrodes need not be inert as the electrode between 1 and 2 may consist of oxidized copper and the electrode between 2 and 3 of lead; in that case the copper electrode is oxidized by the hydroxyl and again reduced by the sodium ion with the production of sodium hydroxide, while the lead electrode is chloridized and again reduced by hydrogen with the production of hydrochloric acid; these products are washed out of the cell by water. The inventor proposes to short-circuit the electrodes at each side of the compartments 2 and 4 as indicated

by the dotted lines 12. "The current then shunts around the bodies of water or other electrolyte in these intermediate compartments. The sodium and chlorine set free in the electrodes, however, with the intermediate body of water, constitute a voltaic cell, the hydroxyl of the water oxidizing the sodium while its hydrogen combines with the chlorine." (1,102,209, June 30, 1914.)

London Notes

The Iron and Steel Institute

The autumn meeting of this Institute will be held in September in Paris in the Hall of the Comité des Forges. Visits to the Basilica of St. Denis and to the works of the Compagnie des Metaux, as well as to Nancy for the purpose of inspecting the metallurgical works and mines, have been arranged.

The Institute of Metals

The autumn meeting will be held on September 10 and 11 at Portsmouth. Arrangements have been made for inspections of the docks and of the works of Messrs. Samuel White & Co., at Cowes.

The Congress of Mining, Metallurgy, Engineering and Economical Geology

The sixth meeting of this congress will be held in London next year and will extend from July 12 to 17. Each of the four sections will conduct independent proceedings.

Market Prices—June, 1914

	£	s.	d.
Aluminum ingots, ton lots	85.	0.	0
Alum lump, loose, per ton	£5.5.0	to	5.10.0
Antimony, Star Regulus, ton	£26.15.0	and	27. 5. 0
Borax, British refined crystal, cwt.	17/6	to	18. 6
Copper sulphate, ton	21.	0.	0
Caustic soda, 70%, ton	16.	0.	0
Copper ore, 10% to 25% unit	10/3	to	10. 9
Ebonite, red, lb.	4.	6	
Hydrochloric acid, cwt.	5.	0	
India rubber, Para, fine, lb.	2.	9 1/2	
Mica in original cases, medium	3/6	to	6. 0
Petroleum, Russian spot, gal.	9 1/4		
Quicksilver (Spanish), bottle	7.	0.	0
Sal Ammoniac, cwt.	2.	2.	0
Sulphate of ammonia, ton	16.15.	0	
Sulphur, recovered, ton	5.10.	0	
Shellac, cwt.	3.	4.	6
Platinum, oz., nominal	9.	5.	0
Tin ore, 70%, ton	£82.0.0	to	84. 0. 0
Zinc, Vieille Montagne	25.	2.	6

Copper opened at £63 and declined till the 8th, £61.10.0, then recovered to £62 (11th), and again fell away to £61.5.0 (17th), and after a slight recovery (18th) to £60.1.3 (26th), showing slightly better on the next few days and closing £61.

Tin opened at £142 and was uneven over the first fortnight between the limits of £137 and £140. On the 16th it went lower, £137.15.0, but hardened slightly on the 23d to £139.7.6. A larger drop brought it on the 25th to £135.5.0, but closes slightly stronger £140.5.0.

Haematite opened at 61s. and has remained quite steady at this price.

Scotch pig opened 57s. 3d. and has been rather weak, showing 57s. 3d. on the 4th and 57s. 1 1/2d. on the 16th, to which prices after recovery to 57s. 6d. on the 9th it again dropped on the 26th, declining to 57s. on the 29th.

Cleveland opened at 51s. 6d. and declined slowly to 51s. 1 1/2d. on the 17th, and again, after slight recovery on the 25th, continuing downward to 51s. on the 29th and closing same price.

Lead opened £19.10.0 and reached £19.17.6 on the 4th, afterward rather lower, but again stronger on the 15th and rose to £20 by the 19th, after which it declined, reaching £19.5.0 by the 29th, but better at the close £19.10.0.

Synopsis of Recent Chemical and Metallurgical Literature

Iron and Steel.

Tilting Open-Hearth Furnaces.—A very interesting series of experiments has been carried out at Witkowitz, Austria, to compare the efficiency of the Talbot furnace with the Wellman tilting furnace and the stationary 69-ton basic open hearth. Dr. Schuster, the general manager of the Witkowitz Iron and Steel Works, gave a detailed account before the semi-annual meeting of the German "Verein deutscher Eisenhüttenleute" of the requirements of these works and of the considerations and deliberations preceding the final decision to build a new open-hearth plant. This plant was to be big enough to take care of their rolling mills and forging presses, capable of extension and yet flexible enough to allow a change of processes in case one or the other of the new furnaces should be no success.

So far England and America had been the pioneers of the tilting open-hearth furnace for continuous operation, whereas continental Europe, aside from a few unfortunate trials, showed no inclination for the Talbot process. As a matter of fact, the pig-iron-ore process promised to be the most economical under the local conditions and circumstances. And although two German experts who had investigated every English Talbot furnace by order of the company came to the conclusion that, only under certain restrictions, it would be advisable to use this process, Dr. Schuster decided to build two Talbot furnaces of 300 tons capacity each, three stationary 60-ton basic furnaces and one 60-ton Wellman furnace, the latter one in order to compare stationary and tilting furnaces of equal capacity. One of the Talbot furnaces was to serve a heatable mixer as the blast furnaces are at a distance of 1.3 miles from the steel plant, and the other should perform the same work in case the experiments should fail. This, however, did not happen, the new installations proved to be most successful and the extension of the plant is planned with Talbot furnaces. It also contains two electric furnaces, one of 2-ton capacity for melting ferro-manganese, the other a 6-ton furnace for special brands of steel.

When working with 5 per cent of scrap in the Talbot, 13 per cent in the Wellman, and 25 per cent in the ordinary open hearth, the first cost of the three furnaces compares 1659:1259:1000. Figured per ton of product, however, this proportion is reversed and reads: 882:1070:1000, which shows the superiority of the Talbot furnace as to cost of erection. With pig iron containing 1.1 per cent of phosphorus, 1 ton of steel is produced in the three types in the same order of rotation in 4 min. 30 sec., in 7 min. 12 sec., and in 8 min. 30 sec. The metallic iron reduced from the additions, of ore is 86 per cent, 80 per cent and 78 per cent respectively, and every quality of steel required is produced as reliably as in the stationary furnace.

Those present at the meeting were provided with extensive maps, tables and cost sheets showing the advantages of the combination as worked out for the special requirements at Witkowitz. They are too widely different from American conditions to be treated in detail here. Those familiar with the art will be interested to know that contrary to the prevailing tendency in building the furnaces a very limited use of water-cooling devices has been made and air cooling applied wherever possible.

During the discussion of the paper it developed that the Königshütte in Upper Silesia, for a few months, has had tilting open-hearth furnaces, a similar process being in operation of which little was known; the re-

sults obtained were so far very similar to those cited above. When speaking at the subsequent banquet, Dr. Schuster remarked that he had been more liberal than customary among the members of this society, in communicating his experiences for the common benefit of all, and hoped that his example would be followed by many successors.

Gold and Silver

Resistivity of Pure Silver, Solid and Molten.—In a paper published in the *Journal* of the Franklin Institute, July, 1914, Dr. Edwin F. Northrup announces the results obtained in determining the resistivity of pure silver at temperatures between 20 deg. C. and 1340 deg. C. The silver used was very pure, being part of the supply used by Prof. G. A. Hulett in his study of the silver voltameter. For determining the resistivity between 20 deg. C. and 60 deg. C., the author used soft drawn wire, and oil-bath, mercury thermometer and Kelvin double bridge. The same silver was then used in a container of magnesite 40 parts and alundum 60 parts. The results were not checked, partly for lack of time and partly for the reason that the author's previous experience with the method makes it seem improbable to him that any serious errors have crept in. The table below gives the results of the determination. At the moment of fusion the resistivity suddenly increases from 5.5 to 16.6 microhms. The measurement was not carried to higher temperatures on account of failure of the thermocouple, due to opening of its circuit.

Degrees Centigrade	Microhms per Centimetre	Ratios, ρ_t/ρ_0	Remarks
20	1.65	Increase per 100° C. = 0.60 microhms.
60	1.90	
960.5	8.4	
960.5	16.6	$\rho_m/\rho_s = 1.98$	Silver solid.
1000	17.01	Silver molten.
			Temperature coefficient calculated from 1000° C. = .00069.
1100	18.19	Increase per 100° C. = 1.177 microhms.
1200	19.36	
1300	20.54	
1340	21.01	

Copper

Proposed New Converter and Converter-Smelting of Ores.—In the June, 1914, *Bulletin* of the American Institute of Mining Engineers, Mr. Herbert Haas publishes a long article describing the construction and proposed operation of a new type of copper converter, in which it is proposed to extend the matte-bessemerizing process to the direct reduction of fine pyritic ore. Converter-smelting is not new, having been practised to some extent at Great Falls and Cananea; but Mr. Haas makes some novel proposals which are intended to widen the scope of the process. His paper is too long to present in more than a very brief abstract.

The general premises on which the author makes his proposal are as follows: The cost of copper converting is now about \$5 per ton of copper, or \$2 per ton of 40 per cent matte. If further economy is to be attained a process must be adopted which will be cheaper than the present established practice of combined blast-furnace smelting and bessemerizing operations. The author explains that his proposed process will not displace existing methods, for converter-smelting is limited to the treatment of ore containing sufficient iron and sulphur to furnish the heat required to produce and sustain the smelting and bessemerizing temperature. The tonnage of concen-

trate that can thus be smelted depends chiefly on their iron and sulphur contents; a decrease in these two constituents in the concentrate requires an increase of them in the matte of the charge. Thus the matte should be low in copper and high in iron and sulphur, in order to furnish the requisite heat to smelt the more silicious concentrate, which would be high in copper but low in iron. The copper concentrate would then take the place of silicious flux to slag the iron in the matte.

As the rate of smelting and the elimination of iron and sulphur depend on the rate at which air is blown into the charge, Mr. Haas has given attention to the design of a converter-smelter that would provide larger volumes of air than are now required. Further, in order to gain technical efficiency the operation must be carried out in such a way as to allow no free oxygen to be discharged in the waste gases. Finally, the air should be used at as low a pressure as possible to reduce the cost of compressing.

Mr. Haas discusses the development of copper converting, and then describes his proposed converter. Briefly, it is a vertical type, bottom-blow converter with elevated tuyere bottom. The shell is lined with magnesite bricks. The most radical departure is the proposed use of a large number of tuyeres of small cross-section, confined to a central area in the converter bottom. Tabulated data are given showing an example of blowing a charge in such a converter. The smelting of fine pyritic ore or concentrate would be accomplished by blowing it into the charge with the compressed air.

The author discusses the air requirements for converting; the cost of power to furnish air, and the thermo-chemistry of the converting process.

Magnetic Separation of Minerals

The Ullrich Magnetic Separator.—An article by Mr. W. Mewes in the *Mining & Engineering Review* (Australia) for May and June, 1914, describes the construction and method of operation of the Ullrich magnetic separator for both wet and dry work, and gives the results of tests. The Ullrich machine has stationary poles disposed about an annular support, and an annular armature rotating above them. Magnetic fields are generated in the intervening space. The armature is divided into a number of concentric rings of V-section, creating in each magnetic field as many zones of action as there are rings to the armature. Ore may be fed either wet or dry, but different constructions are used to meet the two different conditions. The following table gives some practical results obtained with the machines.

Ores	Content, per Cent	Non-magnetic Product, per Cent	Magnetic Product, per Cent	Recovery, per Cent
Tin-tungsten	30 WO ₃ 32 Sn	3 WO ₃ 62 Sn	68 WO ₃ 4 Sn	90 WO ₃ 95 Sn
Tungsten conc.	62 WO ₃ 3 Bi	10 WO ₃ 12 Bi	72 WO ₃ 0.15 Bi	97 WO ₃ 95 Bi
Tin, ilmenite	10.64 Sn	66.83 Sn	0.03 Sn	99.5 Sn
Spathic zinc	15.9 Fe 3.8 Mn 27.8 Zn 49 Zn	30.1 Fe 8.2 Mn 2.5 Zn 75.2 Mn 92.0 Zn
Spathic copper	38.9 Fe 1.01 Cu 3.97 Cu	39.8 Fe 0.13 Cu	68 Fe 88 Cu
Hematite	37.34 Fe 0.238 P	65.32 Fe 0.025 P	80 Fe
Roasted spathic iron	37.2 Fe 6.9 Mn	48.1 Fe 8.5 Mn	93.5 Fe 89.5 Mn
Blast furnace dust	43.0 Fe 0.47 P	57.3 Fe 0.29 P	91.1 Fe

Crushing and Grinding

Grading Analyses by Elutriation.—In reply to discussion on his paper on this subject before the Institution of Mining & Metallurgy, Mr. H. Stadler makes further remarks in *Bulletin* 117 regarding the usefulness of this method of making physical analyses of the finest portion of crushed ore. In view of the policy of fine grinding now in vogue in cyanidation, and of the losses attending the concentration of slime from base-metal ores, the author believes that elutriation should be more widely used for making grading analyses. The method of elutriation begins where screening stops, and besides being more accurate, it is in accord with practical operating conditions, as the handling of slime pulp consists almost exclusively of hydraulic methods.

"We are too easily inclined to consider slime as a definite product, yet the difference between one slime pulp and another may be just as great as between a 10 and 1000-mesh battery pulp. A percentage of moisture or a flow-velocity suitable for one slime pulp may be disastrous to another. The fact that the slimy portion of the ore is responsible for over 80 per cent of the total losses sufficiently warrants a closer investigation by the method of elutriation, which would certainly be a great help in tracing the causes of these losses, and a guidance for the introduction of possible improvements."

The proper operation of hydraulic classifiers can be accomplished only by correct dimensioning, and this, in turn, is involved with the velocity of the rising current of water which carries over the particles of desired size. From this empirically determined velocity and the flow quantities, expressed in fluid tons, the required classifying area may then be calculated by the following formula, in which 16.2 is the conversion factor from the metric to the British standards.

Classifying area in sq. in. =

Fluid ton per 24 hr. \times 16.2

Velocity in mm. per sec.

The following example is given for calculating the diameter of a circular tube mill classifier, by which a 64-mesh battery pulp of 75 per cent moisture is to be classified into a thick underflow of 30 per cent moisture containing solids + 120 mesh, and on overflow containing the — 120 mesh portion:

Intake		
Mesh	Weight	Dry Solids
+ 120	66 2/3	300 tons
— 120	33 1/3	150 "
450 tons of solids, sp. gr. 2.7,	166.6 fluid tons	
water (75% of inlet pulp),	1350.0 " "	
	1516.6 " "	
Overflow		
— 120 mesh, 100%, 300 tons, sp. gr. 2.7,	55.5 fluid tons	
water (89% of overflow pulp),	1221.5 " "	
	1277.0 " "	
Underflow		
+ 120 mesh, 100%, 300 tons, sp. gr. 2.7,	111.1 fluid tons	
water (30% of underflow pulp),	128.5 " "	
	239.6 " "	

The velocity of water effecting a separation similar to that made by 120-mesh screen is 10.08 mm. per sec. Hence:

Classifying area in sq. in. =

$$\frac{1277 \text{ fluid tons} \times 16.2}{10.08 \text{ mm. per sec.}} = 1060 \text{ sq. in.}$$

equivalent to the area of a circular classifier of, say, 3-ft. diameter.

The method of classifying by elutriation is based on different principles than sizing by screens. Neither is an accurate means of measurement, but each has its sphere of usefulness. While sizing by screens is suitable for the study of mechanical efficiency of grinding machines, grading elutriation is more trustworthy for the investigation of metallurgical problems, as it is in accord with condition of actual operation.

Comparative Hardness of Ores in Tube Milling.—

In the April *Journal* of the Chamber of Mines of Western Australia, Mr. W. B. Blyth contributes "a simple and accurate method for comparing the hardness of ore sands judged from a tube milling standpoint." The comparison does not extend to any other crushing machine. Stadler's method for computing the grinding efficiency is used, but the regrind factor is only an arbitrary term used for want of a better. The method is as follows:

"A sample of ore to be tested (wet crushed) is graded (I.M.M. screens) until 8 lb. of a — 20 + 40 sand is obtained. A charge consisting of 8 lb. of sand, 8 lb. of flintstones, 1-in. to 2-in. diameter, and 16 lb. of water, is introduced into a 5-gal. glazed earthenware ginger-beer barrel and ground for 90 minutes at 60 r.p.m. A sample of the ground product is then taken, dried and graded with a mechanical grader, 40, 60, 100 and 150-mesh I.M.M. screens. Grading occupies 20 minutes. The value of the grading in energy units is then calculated by Stadler's method and this amount, minus the original value in energy units, represents the regrinding factor in the table."

Screen	FEED			PRODUCT		
	Mean Factor, Stadler	Per Cent	Energy Units	Per Cent	Mean factor, Stadler	Energy Units
+ 40	17.5	100	1750	16.8	17.5	294.0
+ 60				29.2	20.0	384.0
+ 100				16.4	22.0	360.8
+ 150				6.6	24.0	158.4
- 150				31.0	26.5	821.5
			1750			2218.7

Total energy units, product, 2218.7; feed, 1750; difference, regrinding factor, 468.7.

Cost of Power for Air in Converting.—In discussing this subject (*Bulletin*, A. I. M. E., June, 1914) Mr. Herbert Haas makes the following observations: With a good boiler 1 lb. of coal will evaporate 8 lb. of water; a good steam turbine direct-connected with a turbo-blower will consume 12 lb. of steam per effective hp-hr.; so that 1.5 lb. of coal would produce an effective hp-hr. With fuel oil, an evaporation of 14 lb. of water can be obtained from 1 lb. of oil, so that not quite 0.9 lb. of oil would be required for one effective hp-hr. With Diesel engines a hp-hr. could be produced for less than 0.5 cent, with fuel oil costing \$1.50 per bbl. of 330 lb., and including 12.5 per cent interest and amortization on the capital invested. One cent per hp-hr., however, is taken as the cost of power for purposes of calculation. For converting a 40 per cent ideal matte, the author gives 101.3 hp-hr. as the commercial requirement per ton of copper for furnishing air to the converter.

A New Porous Mineral Filter Medium

In the numerous commercial processes employed by electrochemical, chemical and metallurgical engineers in their practice there is a wide-spread need for a suitable porous medium. For processes requiring a porous diaphragm or filtering medium such materials as canvas, asbestos fiber, loose mineral substances, etc., are resorted to. These at best offer many objections, among which are the necessary and expensive renewals due to the complete destruction or rapid deterioration of the materials used and clogging of the pores, also the interruption of the operation resulting from the frequent cleaning. In many commercial processes the question of filtering thoroughly, rapidly and cheaply is a most vital one and there has been and is a wide search for a material to accomplish this work satisfactorily.

A uniformly porous septum possessing resistance to high temperatures, acids and chemical solutions and as well as showing little liability to clogging of the pores by fine particles, approaches the ideal medium for commercial filtration of gases and liquids. The development of a new porous mineral medium termed "filtros" bids fair to overcome many filtering difficulties encountered in modern industrial fields.

Filtros is composed practically of silica and is consequently inert to all acids (with the exception, of course, of hydrofluoric), the weaker alkalis most commonly requiring to be filtered in commercial practice, and neutral chemical solutions of an organic or inorganic nature. It will also withstand temperatures up to 1800° F.

The method of producing these tile was discovered by Mr. J. E. Porter, of Syracuse, N. Y. Broad patent claims on the various features of the invention have been allowed. These have been acquired by the General Filtration Company, Inc., which company also possesses a number of other patents covering the use of porous mineral septa in many industrial arts. The Harbison-Walker Refractories Company by arrangement with the General Filtration Company, Inc., manufactures the tile for the General Filtration Company, Inc.

Filtros is a white, rigid and porous ware of excellent strength. It is made in the form of standard tile 12 in. x 12 in. x 1½ in. and where this shape can be conveniently applied there are advantages in using this size. A great variety of shapes, however, are possible for devices of special design. The pores of the standard tile are so fine as to offer great resistance to the penetration of very small particles, yet the mass is so porous as to permit the free passage of the gases or liquids to be filtered; in some instances reducing the time of filtering 50 per cent and producing a filtrate of absolute clarity. Tile of other grades of porosity than that of the standard are also produced.

The true porosities of the finer grained tile are about equal to that of the standard but the pores are much more minute, and more numerous. These are capable of retaining finer slimes but do not allow as rapid a passage of fluid as do the coarser tile.

The tests in Table I show how the true porosities compare with the air ratings. The air rating is the number of cubic feet of air per square foot of area per minute that at the given pressure pass through the tile 1½ in. thick.

TABLE I
Cubic feet of air at 2-in. water pressure per cent by void

Grade	water pressure	porosity
5-R	12.0	40.10
7-R	8.4	37.44
5-1	6.3	31.56
5-2	4.1	38.24
5-3	2.0	33.27
5-4	.50	39.55

The resistant property of filtros to high temperatures and acid fumes indicates the probable field for it in bag house practice as well as for the filtration of combustible gases without the loss of sensible heat, which, in many cases, is most desirable.

In problems of cyanidation involving aeration the exceptional uniformity in the porosity of filtros indicates its utility in this connection. The air-rating tests of Table II bear emphasis on this remarkable uniformity of the tile. These, as above, show the amount of air per square foot of area per minute that at the given pressure passes through the standard tile 1½ in. thick.

TABLE II

Cubic feet per minute per square foot at given pressure in inches

Specimen	2 inches	4 inches
1	12.0	24.3
2	12.0	23.8
3	11.7	23.0
4	11.9	24.0
5	12.2	24.5

Further indication of the free flow of liquids through the tile is illustrated by the tests of Table III, which give the amount of water filtered per square foot of area of the standard tile 1½ in. in thickness at the given pressure.

TABLE III

Gallons per minute per square foot area

Head pressure		Temperature of water
5 ft.	1.40	50° F.
5 "	1.39	"
10 "	2.15	"
10 "	2.15	"
15 "	2.61	"
15 "	2.60	57° F.
20 "	3.27	"
25 "	3.42	"

Solubility tests in which filtros tile were immersed in chemical solutions for long periods at various temperatures show the inertness of the tile to acids, weaker alkalies and chemical solutions.

The tests in Table IV representing some commercial filtrations will be of interest.

TABLE IV.—COMMERCIAL FILTRATION.

Solution	Strength per cent	Time immersed	Temp. deg. F.	Loss	Condition unchanged
Sulphuric acid	96	24 hrs.	200	.02	"
Sulphuric acid	96	8 days	"	.023	"
Hydrochloric acid	2/3 of 37	24 hrs.	"	.017	"
Nitric acid	1 1/3 of 70	8 days	"	.016	"
Nitric acid	35	8 days	"	.026	"
Hydrochloric acid	70	8 days	"	.000	"
Water distilled		35 days	"	.110	"
Sulphuric acid	20	35 days	"	.140	"
Hydrochloric acid	20	35 days	"	.080	"
Nitric acid	20	35 days	"	.040	"
Zinc Sulphate	Saturated	20 days	"	.021	"
Mercury Bichloride	Saturated	20 days	"	.042	"
Potassium cyanide	2.5	5 days	"	.09	"
Potassium cyanide	.50	20 days	"	.013	"
Potassium cyanide	.50	7 days	"		
Calcium hydrate	Saturated				

Filtros has been proven by test to resist also the action of organic chemicals as well.

Applications for filtros will no doubt suggest themselves to engineers for a variety of uses, such as porous diaphragms in electrolytic processes and as a filtering medium for gases and liquids in the mining, metallurgical and chemical fields. As there is at present a wide demand for a reliable and strong filtering medium for just such purposes, it will be interesting to watch the career of filtros in commercial practice.

Dry Kilns for Timber Products

While the subject of dry kilns for timber products is of special importance to furniture manufacturers and manufacturers of wood products in general, it should also be of interest to chemical engineers in view of analogy of the process with the drying operations in many chemical processes. For this reason attention may be called to the new catalog of the American Blower Company, of Detroit, Mich., on dry kilns for timber products. (Bulletin No. 20, Series 1. 64 pages, illustrated.)

The problem of lumber drying is the removal from lumber of all moisture, of whatever form, contained in its pores. The sap is not pure water, but contains many substances in solution. All the conifers contain resin. The sap of all trees contains albumen. The ideal effect in the drying process is produced by removing not only the liquid or volatile constituents of the sap, but also those chemical compositions—albumen, starch, sugar, etc.—which by absorbing moisture, after being once dried, may set up fermentation, the initial process of decay in wood.

To produce this ideal effect the drying must commence at the heart of the board and work outwards, the moisture being removed from the surface as fast as it exudes from the pores of the wood. The catalog describes the "moist air" process which fulfills this requirement. The surface of the lumber is kept soft, the pores being opened until all the moisture within has been volatilized by the heat and carried off by the strong circulation of the air. When the moisture is removed from the pores, the surface is dried without closing the pores, resulting in lumber that is clean, soft, bright, straight and absolutely free from stains, checks, or other imperfections.

The catalog describes two general types of moist air kilns. In one the air circulation is obtained by natural means and in the other by mechanical means or forced circulation.

For natural-circulation kilns the "progressive" type is first described which operates on what is known as the "progressive" principle, the green lumber, piled on cars, entering at one end of the kiln, and being gradually moved down through the kiln as the dry lumber is removed at the other end. When green lumber first enters the kiln it is not in a condition to stand a high temperature. For this reason the floor heating coils do not extend back to the green end. Therefore, when a car of lumber is run in, it enters an atmosphere moderately high in temperature and quite moist. Under such conditions the sap within the wood is thoroughly soaked, much the same as if the lumber had been steamed. The gentle heat surrounding it starts the sap toward the surface, the drying taking place first in the center of the board.

As the car moves forward it is gradually brought into a hotter and drier atmosphere. The effect is that the moisture in the surrounding air keeps the sugars, starch, albumen, etc., near the surface, in a semi-liquid state, until all the internal moisture has been extracted. The continually increasing temperature eventually draws the moisture from the pores near the surface, and the surface itself is finally finished off in a practically dry atmosphere.

The lumber entering the kiln does not stand over any steam pipes for several hours (which would cause case hardening), but it is subjected to a mild, moist heat, preparatory to entering a more intense heat and drier atmosphere when in a proper condition to endure it without injury.

The path of the air current is longitudinal, or from end to end, not up through the middle and down the side

walls as in other kilns, which results in the same temperatures and humidity from one end to the other. The cold, wet cars at one end chill the air, causing it to drop down beneath the tracks. The heat generated at the other end by the steam coils creates a partial vacuum beneath the tracks, into which the air rushes from the other end and rising up through the lumber circulates toward the wet end, this operation being continuous.

While with the "progressive kiln" the operation is continuous, it discontinues with the "apartment type" of natural circulation kilns which are especially desirable when the stock used by the manufacturer is required in comparatively small quantities, of varying thicknesses, from a diversity of woods and in varying degrees of dryness.

The second part of the catalog deals with forced-circulation kilns. The forced circulation or blower dry kiln is probably the oldest type of modern kilns. This kiln is a greater success to-day than ever before. It will turn out just as good quality, in just as short a time, as any other type of kiln on the market, and with no greater expense for operation if properly handled. This kiln is recommended especially for drying bent stock, staves, heading, shingles, hoops, lath, tub and pail stock, and it is almost universally used in these particular lines. This kiln gives best results when it is operated on the "progressive" principle; that is, the green lumber piled on cars, enters at one end of the kiln, and is gradually moved down through the kiln toward the blower and the heater, as the dry lumber is removed from that end. When the green lumber enters the kiln it is surrounded by an atmosphere impregnated with moisture gathered from the cars ahead, over all of which the air has passed, and is subjected to a temperature from twenty to forty degrees lower than it eventually has to withstand before leaving the kiln when finally dried. The sap within the wood is thoroughly soaked, much the same as it would be if first steamed, only the effect is milder and perfectly harmless.

The heat starts to volatilize the moisture within the pores, bringing it to the surface, where it is taken up by the air and carried off. In this way the drying takes place first in the center of the board, the dampness of the surrounding atmosphere keeping the pores open, thus preventing case hardening or checking. As the car moves forward it is gradually brought into a hotter and drier atmosphere. The continually increasing temperature, with a corresponding dryness, eventually removes all the moisture, the car finally standing directly over the duct where the air rises hot and dry.

Data are given in the catalog on details of the drying equipment (heating coils, etc.), on drying kilns specialties (structural steel rail supports, trucks, traps, temperature and humidity measuring instruments), and methods of loading dry-kiln trucks.

Colorado metal production for 1913 was valued by the U. S. Geological Survey at \$35,449,298. The production of zinc and gold decreased, but silver, copper and lead increased, as compared with the previous year.

The metal production is itemized as follows: Gold, \$18,146,916; silver, 9,325,255 oz.; lead, 87,620,364 lb.; copper, 7,298,269 lb., and zinc, 119,346,429 lb. Cripple Creek produced 60 per cent of the total gold yield in the state. Leadville produced 79 per cent of the zinc.

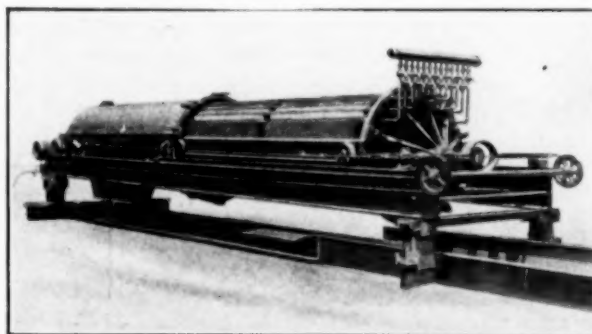
The number of the mines producing metals in 1913 was 858, of which 29 were placers, as against 856 mines, of which 33 were placers, in 1912.

Filter Press Work in Cane Sugar Factories

Owing to the article descriptive of the Worthington filter press work in a cane sugar plant in Cuba, appearing in the July issue of this magazine, and in which certain comparisons were made with a "late type of cylinder press," we deem it only fair to publish the following article for which the Kelly Filter Press Company, of Salt Lake City, Utah, assumes full responsibility.

"The data for the so-called type of cylinder press was taken directly from page 26 of the Kelly Filter Press bulletin entitled 'Filtration Economy in All Industries,' and on account of the fact that the Kelly press is the only late type of cylinder press installed in cane sugar plants so far as can be determined, it is thought only fair to make correction in the article which appeared in the July issue of this paper.

"First and foremost, the data taken from the Kelly Filter Press bulletin was quoted incorrectly. In Table 2 of the article it was stated that the per cent of sucrose in the press cake dumped from the Kelly press was 8 per cent. *This should have been eight-tenths of 1 per cent.* The cake dumped from the Worthington press according to this article contained 6.85 per cent sucrose. In other words, 1 ton of cake dumped from the Kelly press contained 16 lb. sucrose, while 1 ton of cake dumped from the Worthington press contained 137 lb. of sucrose. The sugar saved by the Kelly press over



KELLY FILTER PRESS IN SUGAR PLANT

the other type amounts to a very large saving, as a glance at the following will show.

"Assuming a cane sugar factory grinding 1000 tons of cane per twenty-four hours, and that in such a factory the weight of filter cake dumped approximates 1.5 per cent on the weight of cane crushed, there would be 15 tons of filter press cake dumped per twenty-four hours from plate and frame presses.

"In two separate tables in the previous article it was stated that the cake dumped from the Worthington press contained 6.85 per cent sucrose. This would be, therefore, 1.03 tons, or 2060 lb. of sucrose per twenty-four hours. An equivalent amount of cake dumped from the Kelly press contains only 0.8 per cent sucrose, which would equal approximately 0.2 ton, or 400 lb., of sucrose per twenty-four hours. The difference is 1660 lb. of sucrose in favor of the Kelly. Assuming that 92 per cent of this can be recovered as 96 deg. sugar, then 1527 lb. of sucrose is saved every twenty-four hours by Kelly filter-press methods. It will cost, say, $\frac{1}{4}$ cent per pound to make this into sugar. If sugar is worth $2\frac{1}{4}$ cents per pound, then we can figure the actual value of sugar per pound at this station at 2 cents. *This means a sucrose saving per day of \$30.54, which on a 200-day campaign would equal \$6,104.*

"It is in savings of this nature and magnitude by which the Kelly press has proven its worth as an efficient filter.

Length of Cycle

"In the previous article in Table 2, it is stated that the average of the Worthington cycle is forty-five minutes and that of the Kelly press is 160 minutes. This gives an incorrect impression due to the fact that a period of wash was used in the Kelly press cycle, and evidently no wash was used in the cycle of the Worthington press.

"The cake dumped from the Worthington press per cycle is given at 179 lb., containing 58.42 per cent and 6.85 per cent sugar. The remaining portion of the cake would therefore represent 34.73 per cent of the weight which would be the dry insolubles. The amount of solids would therefore be 62 lb. per cycle. According to our data sheet, there are approximately 279 lb. of dry insolubles per cycle dumped from the Kelly press. The filter area of the Worthington press is given as 300 sq. ft., while that in the Kelly press is 420 sq. ft. This shows a capacity of 0.207 lb. of cake per square foot of filter area per cycle in the Worthington press, as against 0.66 lb. per square foot of filter area per cycle in the Kelly press.

"It seems to be a predominant opinion among cane sugar men that it is absolutely essential to have a washing machine in connection with the filter press department for washing the filter bags. We cannot emphasize too strongly the fact that we do not require a washing machine in connection with the Kelly press because the cloths are washed thoroughly inside of the press.

"The claim is made the same set of bags was used continuously until worn out. They are not removed from the press for washing. The permeability of the cloths is restored each cycle, thereby constantly maintaining high efficiency in the ratio of filtration of the cachaza.

"Another feature of the Kelly operation which we wish to emphasize is the fact that the sugar is washed out of the filter press cake directly in the press, and that such wash is practically perfect, the sugar being virtually pushed out of the cake in place; or, in other words, displaced directly with water. This means a minimum dilution of the sugar juices. It has been proven that this dilution is negligible as compared to that resulting from the addition of water to the cachaza before filter pressing, or the dilution accompanying the disintegration of filter cake with water followed by a secondary filtering operation, as sometimes practiced in connection with the plate and frame presses. The latter, we assume, is the method of obviating sucrose losses in the Worthington press.

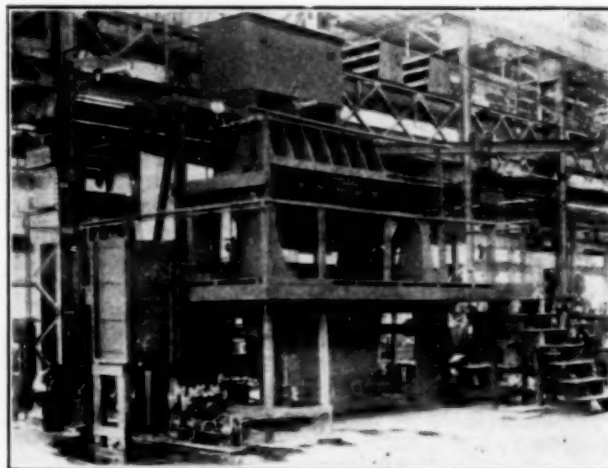
"We would be very glad to compare the brightness of the juices after filtration by the Kelly press; life of filter bags; labor required; the heat conserved; the leakage, breakage and maintenance costs and repairs with that obtained by any other filter apparatus on the market.

"The data above submitted for the operation of the Kelly press have been taken from tests made under the supervision of the chemist of the Adeline sugar factory where eight Kelly presses are installed. Those readers desiring more complete detail on the above data, may be referred to insert at page 26, of the Kelly Press Company's bulletin, which will be furnished upon request by that company."

Materials of engineering construction will be one of the subjects considered at the International Engineering Congress, 1915. Among the topics that will be covered will be timber preservation, clay products, concrete, special steel, copper and alloys, aluminum, and the testing of materials. Full information can be had from the Congress, Foxcroft Bldg., San Francisco.

Filter Press Laboratory

In a recent issue we described the Worthington press as installed in a Cuban sugar central. It is to be noted that this invention and improvement is one that will materially aid this particular industry. The press is, however, suitable for many other purposes such as dewatering of slimes. Methods developed in this press bring out the possibility of handling colloids without the introduction of any unnecessary amount of sand, inasmuch as it does not force through the cake or the filtering medium, obviating closing of same. The use of this press in handling wool grease or sludge, as well as sewage and tankage, has been fully demonstrated and accomplished. Its importance in handling clay products rapidly and continuously has been proven. It is a self-contained construction with easy accessibility and gives many advantages, such as quick and efficient closing and opening for charging and discharging, quick dropping of cake by gravity, manual labor cut down to the smallest, and power consumption hardly any. Such simplification and easy manipulation, saving in filter cloth and greater recovery in either clear liquors or clean press cakes, gives an improvement that means reduction in cost of operation.



WORTHINGTON FILTER PRESS LABORATORY

The illustration shows the laboratory of Henry R. Worthington, Harrison, N. J., where a 6-ft. x 9-ft. press is in operation for experimental purposes, all completely fitted up with agitator tank, filling pipes, supply pump for material to be filtered and small pressure pump producing final squeeze. The press shown has automatic closing features operated by water pressure and self-dumping bag arrangement for dropping cake in a belt conveyor. There is maintained a complete chemical and experimental laboratory, where all filtration tests are made of material sent by prospective customers. They have a corps of efficient and practical filtration experts and engineers, who install and start these presses to get the best results.

Brass-Furnace Practice is the title of Bulletin 73, by H. M. Gillett, recently issued by the U. S. Bureau of Mines. It contains a report into the losses sustained in melting nonferrous alloys, which amount to about 5 per cent of the original metal. It is shown that from 90 per cent to 95 per cent of the heat units in fuel used in melting do no useful work. The financial saving to be effected by improved practice in melting, due to reduced loss of metal, increased efficiency of fuel and crucible life, is estimated to be about two million dollars.

Hydraulic Reducing Valve

The adjoining illustrations show a new type of hydraulic reducing valve which the Auld Company, 1247 North 12th Street, Philadelphia, is just putting on the market. The illustration shows a 4-in. valve for an inlet pressure of 800 lb. and an outlet pressure of 500 lb. The construction of the valve is simple and all parts are strong enough to insure that it will give continuous and reliable service. The outlet pressure does not vary, though the volumes of water being passed through the valve may vary. The operation of the valve is as follows:

The initial pressure enters by the branch marked "inlet" and acts between the main valve 7 and the piston 4. The piston is of greater area than the valve 7 and therefore the main valve is held tightly shut on its seat 8. To prevent water from passing from the top to the bottom side of the main piston 4 a cup leather 6 is fitted.

The initial pressure passes from the high-pressure side into the port *B'* which communicates with the space at the top side of the regulating valve 13. This regulating valve is held open by action of the spring 23 and the high pressure is allowed to pass through the valve 13 and down the pipe 22 into the chamber under the piston plate 5.

With high pressure acting on both sides of the piston 4, the valve 7 is forced open, due to the area of piston 4 being greater than the valve 7, and pressure flows into the outlet or low-pressure side. The low-pressure side of the main valve body 1 is connected to the diaphragm chamber of the regulating valve by port *A'* and when

chamber and through the port *A'* into the low-pressure side of the main valve body. This action is repeated so long as the pressure on the outlet side tends to vary.

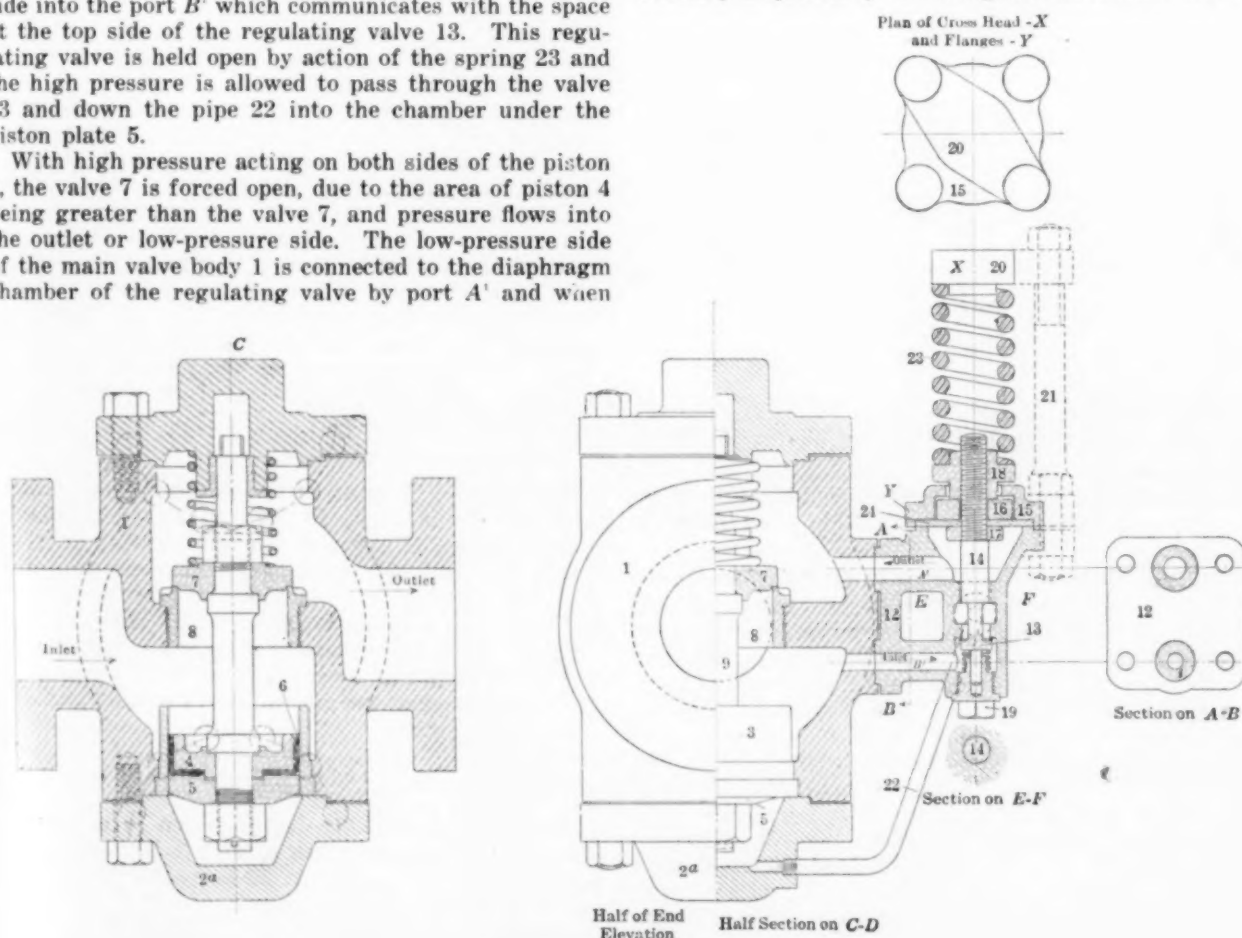
A slight variation of the outlet pressure is sufficient to operate the reducing valves and any fluctuations which take place are thereby rendered negligible.

Personal

Dr. Lucien I. Blake, well known for his inventions in the field of electrostatic ore separation, is delivering a series of lectures on Cosmic Physics at the University of California. He will spend some time in Denver after concluding his engagement about August 1.

Dr. Leo H. Baekeland, of Yonkers, N. Y., started on July 11th on a trip around the world. He is now en route to Japan and Korea.

Mr. Albert Burch will have general management of the Aurora Consolidated mine and mill which was recently acquired by the Wingfield interests. He will



HYDRAULIC REDUCING VALVE

the pressure under the diaphragm acting on the area of the piston 16 exceeds the load which the spring 23 is set at it forces up piston 16 and valve 13 stopping the flow of water to the underside of the piston 4 (i.e., chamber under piston plate *S*) and allows the main valve 7 to come to its seat and stops the flow of pressure from the inlet side.

There is a constant leakage from the underside of piston 4 to the low-pressure side through the slots cut in the body of the regulating valve (shown at section on *EF*) and when the regulating valve 13 closes, the surplus water under the piston 4 is forced out through the pipe 22, through these slots into the diaphragm

retain also general management of the Goldfield Consolidated. Mr. K. M. Simpson has been made assistant general manager of the latter company.

Mr. J. M. Callow, of the General Engineering Co., Salt Lake City, Utah, is supervising the installation of the 200-ton experimental flotation plant for the Inspiration Consolidated Copper Co., Miami, Ariz.

Mr. Fred Close, metallurgical engineer of London, has returned from a professional trip to Alaska.

Mr. J. V. N. Dorr received the honorary degree of Mining Engineer at the recent commencement of Rutgers College, in recognition of his work in the metallurgy of gold and silver.

Mr. R. C. Gemmell, general manager of the Utah Copper Co., was chairman of the organization committee having in charge the establishment of a branch of the American Institute of Mining Engineers in Salt Lake City. The organization meeting was held August 13.

Mr. W. G. Haldane, acting president of the Colorado School of Mines, was granted the degree of Doctor of Science at the commencement exercises of Denver University.

Mr. H. W. Hardinge is on a business trip through Canada, Alaska and the western United States.

Mr. D. C. Jackling, managing director of the Utah and other copper companies, has returned from his Alaskan trip. He expects to go East for conference with his associates and will then return to Alaska.

Mr. J. W. Jeffrey was recently elected second vice-president of the Jeffrey Manufacturing Co., Columbus, O.

Mr. C. D. Kaeding, formerly assistant general manager for the Goldfield Consolidated Mines Co., Nevada, has been appointed vice-president and general manager for the Dome Mines Co., Ltd., at South Porcupine, Ontario.

Mr. F. A. Kaufmann has resigned his position with the Tottenville Copper Company and is now connected with the Roessler & Hasslacher Chemical Company at Perth Amboy, N. J., as chemical engineer.

Mr. Guy M. Kerr, formerly superintendent of the Arkansas Valley plant of the A. S. & R. Co., Leadville, Col., has been transferred to an executive position at the Garfield smelter, near Salt Lake City. He is succeeded at Leadville by Mr. J. F. Austin, formerly superintendent of the same company's plant at Monterey, Mexico.

Mr. William D. Leonard has been appointed superintendent of the Garfield Smelting Works, Garfield, Utah.

Mr. Bernard MacDonald has removed his office from Pasadena to 533 I. W. Hellman Bldg., Los Angeles, Cal.

Dr. Harold Pender, professor of electrical engineering at the Massachusetts Institute of Technology and director of the research division of the department of electrical engineering, has been invited to become professor-in-charge of the department of electrical engineering at the University of Pennsylvania, and has accepted this invitation, which will become effective next fall. Heretofore the departments of mechanical and electrical engineering at the University of Pennsylvania have been under the joint direction of the Whitney professor of dynamical engineering. From the beginning of next year the department of electrical engineering will be on an independent and coordinate footing with the departments of civil and mechanical engineering.

Dr. Edward D. Peters was given the honorary degree of Doctor in Engineering by the Royal School of Mines, Freiberg, Saxony.

Mr. William C. Post has resigned his position as metallurgical engineer with the Illinois Steel Co., and will take a similar position with the Jeffrey Manufacturing Co., Columbus, O.

Dr. Rossiter W. Raymond delivered the commencement address at the Colorado School of Mines. While in Denver he was the guest of honor at a dinner tendered by the local scientific and mining men.

Mr. P. A. Robbins, general manager for the Hollinger Gold Mines Co., Timmins, Ontario, has returned from a trip abroad and resumed his duties.

Prof. Robert H. Richards has been making a tour of the Western metallurgical centers gathering data on ore dressing. The itinerary includes Lake Superior, Montana, Idaho, California, Nevada and Utah.

Mr. Robert C. Sticht, general manager of the Mt. Lyell Mining & Railway Co., Tasmania, is on an extended vacation which will take him to Germany. On his trip through the United States he has visited the scenes of his former activities in smelting in Montana and Colorado. While in Denver, Mr. Sticht was the guest at a dinner given by the Colorado Scientific Society, of which he has been a member for over 30 years.

Mr. L. C. Turnock has resigned his position with the Pennsylvania State College as head of the Department of Electrochemical Engineering, which he has held for the past three years, and has accepted a position with the Edison Storage Battery Company in Orange, N. J.

Obituary

We have been advised of the accidental death on June 29 of Arthur Austin, chief chemist for the International Smelting & Refining Co., at Tooele, Utah. He was engaged in testing work and had been experimenting with oil burners. A steel oil-container under high pressure exploded, covering him with oil which immediately caught fire. He was so severely burned that he died a few hours later. He leaves a widow and young child. He was a son of L. S. Austin, a graduate of the Colorado School of Mines, '05, and had been in the employ of the Anaconda Copper Mining Co. practically since his graduation. He was transferred to the Tooele works several years ago. In his work with these companies he had given excellent satisfaction, and showed promise of developing into a competent metallurgist.

Benjamin F. Thomas, of Chattanooga, Tenn., died in Atlantic City, N. J., on June 25, at the age of fifty-two years. He was ill only a few months, having suffered a stroke of paralysis in January. He was one of the founders of the Coca-Cola Company and president of the Coca-Cola Bottling Company. Mr. Thomas was interested in many departments of technical chemistry and will be remembered by our readers as a constant and jovial attendant at the meetings of the American Electrochemical Society.

Notes

A patented substitute for platinum to be used for electrical contacts consists of an alloy of 45 per cent platinum, 15 per cent gold, 25 per cent silver and 15 per cent copper.

Stratton's Independence, Ltd., Cripple Creek, Col., milled 11,800 tons of low-grade mine and dump ore in the month of May, 1914, yielded a net working profit to the company from both mine and mill departments of \$10,150.

Stamp duty on the Rand is steadily increasing. In 1908 it was 6.27 tons per stamp in 24 hours. In 1909 it was 6.79; in 1910, 7.19; in 1911, 7.88; in 1912, 8.39; and in 1913, 8.82. It is anticipated that the duty for 1914 will average 9 tons or over.

Transvaal Gold Production.—The number of companies reporting to the Transvaal Chamber of Mines in April, 1914, was 60. The total quantity of ore milled during that period was 2,130,545 tons. There were 9695 stamps in operation, with an average duty of 8.85 tons per stamp in 24 hours. Tube mills in

commission numbered 290. The yield for the month was 683,877 fine ounces of gold. Compared with the report of a year ago the current report shows 350 fewer stamps dropping, but 5 more tube mills in commission. The current yield is less than that of April, 1913, by 101,097 ounces gold. The yield per ton milled is about the same as a year ago.

Steel Chain Belt.—The Chain Belt Company, of Milwaukee, Wis., have issued their illustrated catalog No. 59 on Chabelco steel chain belt. This is an all-steel, case-hardened-bearing roller chain belt designed to meet the most difficult conditions. The side bars are punched from mild-steel bar stock. The rollers for transmission purposes are made of the best grade of cold-rolled steel shafting, machined out of the solid on automatic turret lathes. The rollers for elevating and conveying purposes are made of cold-rolled steel, cast steel or grey iron. The bushings are stamped out of cold-rolled pickled steel. The pins are made of forged steel and milled flat on ends to fit oblong hole in side bar to prevent turning. Every pin and bushing is case-hardened and heat-treated. Chabelco steel chain belt can be used by practically every industry for elevators and conveying and transmission purposes and has been introduced with particular success in pulp and paper mills, power houses, canning plants, cement mills, fertilizer plants, packing houses, ice plants, oil wells, sand and gravel plants, glass plants and potteries. Chabelco steel chain belt is also used on the same company's pivoted bucket systems of both overlapping and contact-lip type; the capacities of these systems range from 20 tons per hour up. A special study of methods for handling coal and ashes has also been made by the Chain Belt Company. The catalog is well illustrated and gives details of types, sizes and dimensions.

Secondary metals recovered from waste and scrap, exclusive of gold, silver, platinum and iron, in the United States in 1913, had a gross value of \$72,845,000. After remelting and refining these secondary metals sell at only slightly lower prices than new metals. For purposes requiring metal of exceptional purity it is necessary to use virgin metal, but otherwise the secondary metals can be used in whole or in part. The use of magnetic separation in the recovery of secondary metals is on the increase. Following is a tabulation of secondary metal production, prepared by the U. S. Geological Survey:

Production of Secondary Metals in the United States in 1912 and 1913

Metal.	1912		1913	
	Short tons.	Value.	Short tons.	Value.
Secondary copper, including that in alloys other than brass	66,441	\$21,593,325	66,980	\$20,536,068
Remelted brass	161,523	27,279,516	99,315	24,651,969
Secondary lead	30,266	6,045,120	33,104	6,409,392
Recovered lead in alloys	36,902		39,730	
Secondary spelter	32,251		50,005	
Recovered zinc in alloys other than brass	3,912	7,750,494	3,743	6,019,776
Secondary tin	8,333		6,415	
Recovered tin in alloys	7,068	14,301,368	7,763	12,567,379
Secondary antimony	13		45	
Recovered antimony in alloys	2,493	426,020	2,660	460,932
Secondary aluminum			2,198	
Recovered aluminum in alloys			2,456	2,199,480
Total value		\$77,395,843		\$72,844,996

The production of primary refined lead in the United States in 1913 amounted 462,460 short tons, being a decrease of 18,434 tons, or 3.8 per cent., compared with the output of 1912. The production of secondary lead was 72,834 tons.

Digest of Electrochemical U. S. Patents

PRIOR TO 1903.

Arranged according to subject-matter and in chronological order.

Compiled by Byrnes, Townsend & Brickenstein, Patent Lawyers, National Union Building, Washington, D. C.

Electroplating (Continued)

453,355, June 2, 1891, Alexander Elliot Haswell and Arthur George Haswell, of Vienna, Austria-Hungary.

Relates to coating metals, such as iron or steel, with an anti-corrosive deposit of binoxide of lead or of a mixture of binoxide of lead with binoxide of manganese by means of a galvanic current. The bath contains one liter of water, eight to eighty grams of nitrate of lead and twenty to fifty grams of nitrate of ammonia. The article to be coated is connected to the positive pole of a source of current and immersed in said bath. In practice, good results have been obtained with a current strength of one-half ampere to one ampere per square decimeter surface of the article to be coated, and a tension of two volts to two and one-half volts. The temperature of the bath may vary from 64 deg. to 75 deg. Fahr. A suitable deposit is obtained in about ten minutes. To the above bath may be added about half a gram of nitrate of protoxide of manganese; this will give a deposit of binoxide of manganese along with the binoxide of lead.

476,639, June 7, 1892, Alexander Elliot Haswell and Arthur George Haswell, of Vienna, Austria-Hungary.

Relates to forming deposits of oxides on metals by electrolysis, and is an improvement on their earlier patent No. 453,355, dated June 2, 1891.

For preparing the improved bath, eighty grams of nitrate of lead are dissolved in five hundred grams of water and this solution poured into six hundred and thirty-five grams of caustic soda solution, having a specific gravity of 1.27, equal to 31 deg. B., and containing one hundred and fifty-eight grams caustic soda. After mixing ten grams of finely powdered carbonate of manganese are added and the bath well stirred during electrolysis. The article to be coated is connected to the positive pole and immersed in the bath. In practice, good results have been obtained with a current of one-half to one ampere per square decimeter of surface to be coated, and a tension of from two to two and one-half volts. The temperature of the bath may vary from 64 deg. to 75 deg. F. A deposit of suitable thickness may be obtained in about ten minutes.

509,124, November 21, 1893, Hannibal Goodwin, of Newark, N. J.

Relates to electro-etching of plates, etc., and describes several well-known methods of producing a design, etc., upon a plate by photographic means. The metal to be dissolved, or etched, is finally exposed, the plate suitably waxed and connected as an anode, and etched in any suitable electrolyte. A plate to be etched and a cathode are suspended in a cell, and connected in series to another plate and cathode in another cell, a number of such plates so connected are electrolyzed by a dynamo current, instead of electrolyzing plates separately by a battery current.

514,896, February 20, 1894, Craig Ritchie Arnold, of Sharon Hill, Pa.

Relates to processes of coating metals with oxides, the article to be coated being connected as anode in a solution of the metal to be deposited as oxide. For depositing peroxide of lead, the anode is immersed in a solution made as follows: to prepare twenty-two gallons, dissolve twenty-two pounds of nitrate of lead in seven gallons of hot water; this solution is poured into eleven gallons of caustic soda of 1.20 sp. gr., after hav-

ing been warmed to 70 deg. C. After about thirty minutes, and when cool, sufficient water is added to bring it to the required volume. During plating, finely divided litharge is added to the solution, and thoroughly agitated. The cathode is placed in a porous cup containing a concentrated solution of caustic soda, either alone, or in combination with bichromate of soda; caustic soda is preferred, although bichromate of soda may be used as the equivalent.

517,275, March 27, 1894, Hannibal Goodwin, of Newark, N. J.

Relates to the decoration of gold or silver by electro-etching and oxidizing or depositing. A suitable design is applied to a gold or silver surface by any of the usual means; the gold or silver is then connected as an anode and suspended in a solution containing iodine, preferably potassium iodide containing free iodine. The electrolysis produces a quick and perfect etching of the unprotected parts of the plate without disintegrating the resisting matter applied for insulation. Since it is very difficult to apply a different color to gold by oxidation, patentee deposits silver upon the etched surface of the gold, and then oxidizes or platinizes the silver; or any other contrasting metal may be substituted for the silver.

526,114, September 18, 1894, Emile Placet and Joseph Bonnet, of Paris, France.

Relates to the electrodeposition of chromium, from aqueous and fused baths; also to the deposition of chromium alloys. For the aqueous solutions, chromic acid or a chromic salt is dissolved in water, to which may be added an acid, and also one or several of a plurality of substances which vary the tint of the deposited metal. For alloys, a salt of the metal to be added is dissolved in the electrolyte. The patent should be consulted for details.

534,259, February 19, 1895, Pompeo Garuti, of Florence, Italy.

Relates to an apparatus for the electrolysis of water, and preferably consists of a lead-lined tank in which is placed an inverted box-like compartment divided into a plurality of separate narrow cells by diaphragms. Anodes and cathodes are placed in alternate cells, the tops of the latter having openings or vents to permit the oxygen or hydrogen to pass into collecting tubes. The electrolyte is preferably pure water containing twelve per cent. of its weight of commercial sulfuric acid of 60 deg. With an acid electrolyte, it is preferred to make the entire apparatus of lead. Instead of acid, a 15 per cent solution of caustic soda or potash may be used, in which case the apparatus may be made of cast or wrought iron, or steel.

534,895, February 26, 1895, William H. Legate, of Hartford, Conn.

Relates to a method of and apparatus for marking and ornamenting tableware, such as knives, etc., and other articles. The article to be marked is coated with a suitable varnish; a metal stamp bearing the design to be imprinted is moistened with strong potash solution and pressed against the varnished surface. The potash dissolves the varnish, exposing the metal in the outline of the design; the article is now washed with water, and then in solution of ammonium chloride is applied to the exposed metal. The knife or other article is now connected as an anode, and the metal stamp as cathode; the electric current etching the article and fixing the mark or design. The article is then washed in potash to remove the varnish, and may be further treated as desired. The apparatus consists of a frame to hold a plurality of knives, etc., and a plurality of metal stamps, in fixed relation to each other.

(To be continued)

Book Reviews.

Organometallic Compounds of Zinc and Magnesium.

By Henry Wren, Ph.D., 12mo. (12 x 15.5 cm.), 100 pages; price \$0.75 net. New York: D. Van Nostrand Company.

A monograph which will give the organic chemist a good résumé of the Grignard reactions with magnesium salts and the corresponding zinc salt reactions. It is particularly concerned with the use of these reactions as intermediaries in isolating difficult organic compounds. It has practically no interest for the metallurgist or electrochemist, since the properties of these organic metallic compounds are scarcely mentioned, and their possible uses in and of themselves ignored. The illuminating and suggestive book on organic-metallic compounds has yet to be written.

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The American Fertilizer Hand Book.—Seventh Annual Edition: 1914. 456 pages. Price, \$1.00. Philadelphia, Pa.: Ware Bros. Company.

The 1914 edition of the well known and useful annual Fertilizer Hand Book contains the usual statistical matter, brought up to date, on the fertilizer trade, the cotton seed oil industry, packing and rendering plants, etc., together with a number of specially prepared annual reviews. Among these we note the following: "The Sulphuric Acid Industry," by Andrew M. Fairlie; "Available Phosphates by Furnace Treatment," by James H. Payne; "Available Nitrogen in Compounds," by E. P. Verner; "Present State of the Cyanamid Industry," by E. J. Pranke; "The World-Wide Search for Potash," by Harry Wilson; "Mining of Florida Pebble Phosphate Rock," by Thomas M. Peters; "The Products of Cottonseed," by Thos. C. Law; "The Use of Packing-House By-Products," by J. B. Sardy.

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The Rolling Mill Industry. By F. H. Kindl, Engineer. Octavo, 76 pages, 27 illus. Price, cloth binding, \$2; flexible leather, \$3. Cleveland, Ohio: The Penton Publishing Company.

According to the title page, this is "a condensed general description of iron and steel rolling mills and their products," and according to page 1, the author has in view to furnish those engaged in the iron and steel business with "a comprehensive knowledge pertaining to this important branch of the iron and steel industry."

The book fails absolutely in its avowed purposes. It is written as if for children under ten years old. The German word "*kindlich*" describes it exactly—with apologies to the author for borrowing his name to describe his book.

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Electric Furnaces for Making Iron and Steel. By Dorsey A. Lyon and Robert M. Keeney. 142 pages, 36 illustrations. Bulletin 67 of Bureau of Mines, Washington, D. C.

This pamphlet of the Bureau of Mines is an excellent and timely summary of the history of electric iron ore reduction and electric steel refining and of the present status of the art. The first chapter, by Dorsey A. Lyon (p. 7 to 57), deals with the electric furnace in pig iron manufacture; the second chapter, by Robert M. Keeney (p. 58 to 137), with the electric furnace in steel manufacture. All furnace designs which have achieved any importance are concisely described and illustrated, together with a few typical installations. Some figures of cost are added. This is a decidedly useful piece of work of the Bureau of Mines.